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Bulletin 160 Mineral Technology 22

DEPARTMENT OF THE INTERIOR.
FRANKLIN K. LANE, Secretary
BUREAU OF MINES
VAN. H. MANNING, Director

ROCK QUARRYING FOR CEMENT MANUFACTURE

BY

OLIVER BOWLES

WASHINGTON
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PREFACE.

Under a cooperative agreement between the Bureau of Mines, the United States Geological Survey, and the United States Bureau of Standards, a study of the stone-quarrying industry of the country was begun in 1914. As a result of the investigations conducted, a series of reports dealing with various phases of stone quarrying are being published by the Bureau of Mines. The first publication, "Safety in Stone Quarrying," Technical Paper 111, appeared in 1915, and was followed by "The Technology of Marble Quarrying," Bulletin 106, in 1916, and by "Sandstone Quarrying in the United States," Bulletin 124, in 1917. Because of the wide demand for the reports already published and for others of similar type, additions to this series will be made as rapidly as the necessary data can be prepared. The present report deals with the quarrying of rock for cement manufacture.

The Portland cement industry has attained such magnitude in the United States during recent years that a report covering the quarrying of the raw materials of cement will, it is believed, be of value to the industry. The enormous increase in the use of concrete for buildings, roads, bridges, dams, retaining walls, and many other purposes has resulted in a vast production of cement. About the years 1907 and 1908 such heavy demands were made upon quarries producing the raw materials that the introduction of large and powerful quarry equipment became imperative. Since then the wide use of such equipment has revolutionized quarry methods. Consequently this is an opportune time to investigate efficiency and safety under modern conditions of operation.

Recent rapid advances in the construction of concrete ships undoubtedly gives added impetus to the cement industry, and tends to increase further the demand for greater efficiency in quarry operation.

Mr. Bowles personally visited 64 of the largest cement plants in the United States, from Oklahoma eastward. The results of his investigations comprise the major part of this bulletin.

In the Bureau of Mines bulletins dealing with the quarrying of marble and sandstone, the necessity for careful blasting is strongly emphasized, as the production of sound blocks is desired. In quarrying rock for cement manufacture, however, the rock must be broken into fragments that can be easily handled, hence it is important to obtain a maximum of fracturing with a minimum of explosive. Effective blasting is, therefore, one of the most important subjects dealt with in this publication. It involves the diameter, distribution, and depth of drill holes, size and nature of the charge, method of firing, quarry conditions that influence blasting methods, and various other factors. Mr. Bowles presents many examples of actual blasting results and their controlling conditions.

During recent years high wages and the scarcity of labor with a greatly increased demand for cement have resulted in important modifications in general quarry methods. The modern tendency is to use labor-saving machinery and methods wherever possible. As a result the method of drilling small holes close together, shooting down the rock in successive benches, and loading by hand has in most quarries given way to steam-shovel loading of rock thrown down in great quantities by large blasts in churn-drill holes.

Certain operators who still retain tripod-drilling and hand-loading methods claim that changing to churn drills and steam shovels would require too great an outlay of capital. The facts brought to light in this investigation indicate that as a general proposition, if the necessary capital is available, it is more economical to scrap the old equipment and purchase modern machinery than to attempt by antiquated methods to compete with modern plants.

Other cement-plant operators having quarries of variable rock claim that the maintenance of a proper mixture requires tripod drilling on low benches and selective hand loading of the rock. By considering every phase of this problem Mr. Bowles makes it clear that in most instances this claim is invalid. Through the application of methods now in successful operation, or a combination of these methods, a uniform mixture may be maintained and the most modern quarry methods be employed.

Methods of transportation within quarries, and from quarries to crushing plants, are discussed in detail, and many improvements are suggested. The chief features reflecting inefficiency are: Needless complication of trackage, especially in quarries with multiple benches; haulage on excessively heavy grades; the combination of too many different systems of transportation in a single quarry; and excessive rehandling of rock.

No attempt is made to discuss problems connected with manufacturing processes except insofar as they are related to quarry methods. The method of mixing the raw materials in order to obtain the proper proportions of the constituents used has, in many instances, a marked effect on the quarry method, and consequently this particular phase of cement manufacture is discussed at considerable length.

Many cost figures, obtained through the kindness of quarry operators, are given, with the principal controlling conditions, to aid quarrymen in judging the efficiency of the methods they use as compared with results obtained by others.

In the type of quarries considered in this publication, quarrymen are subject to many dangers. The discussion of safety problems is not confined to one separate chapter, but wherever machines or methods are discussed the dangers involved are considered and the best means of minimizing such dangers are pointed out.

CHARLES L. PARSONS,
Chief, Division of Mineral Technology.

ROCK QUARRYING FOR CEMENT MANUFACTURE.

By Oliver Bowles.

INTRODUCTION.

As the preface states, this bulletin is the fourth of a series of reports by the Bureau of Mines on different phases of quarrying in the United States. The first part of the bulletin describes the chief types of cements, the growth of the cement industry in this country, and the character of the raw materials used. The bulk of the report deals with quarrying methods and equipment, and gives especial attention to drilling and blasting; a chapter on rock mining and one on prospecting are included. Methods of manufacture are mentioned briefly, as that subject, except as related to quarrying, lies outside the scope of this report; but transportation methods and the effects of the manner of mixing the materials at the quarry or crushing plant on quarrying methods are discussed.

ACKNOWLEDGMENT.

Acknowledgment is hereby given of the cordial cooperation of quarrymen during the field investigations preliminary to the preparation of this bulletin. Special mention should be made of the valuable assistance rendered by Mr. F. J. Leonard, of the Pennsylvania Trojan Powder Co., of Allentown, Pa., and by Mr. S. R. Russell, of E. I. du Pont de Nemours & Co., of Wilmington, Del. Others who rendered noteworthy service are S. S. Smith, of Ironton, Ohio, and R. H. MacFetridge, of Fordwick, Va.

HYDRAULIC LIMES AND CEMENTS.

The presence of lime mortar between the stones forming the ruins of an ancient temple on the island of Cyprus indicates that even primitive man recognized the advantage of using some type of binding or cementing material in building stone structures. Lime mortar, as far as is known, was the first material to be used for this purpose. One by one other types of mortars and cements were invented.

The chief types of cements now used are hydraulic lime, natural cement, puzzolan cement, and Portland cement.

HYDRAULIC LIME.

Hydraulic lime is the product of burning an impure limestone that carries so much lime carbonate in proportion to its silica and alumina content that the burned product will contain considerable free lime in addition to the silicates and aluminates formed. Hydraulic limes, as a rule, contain 75 to 80 per cent calcium carbonate and 20 to 25 per cent clay. The rock is calcined at a temperature slightly above that of decarbonization, and under these conditions silicates, aluminates, and ferrites of lime are formed. The burned product must contain enough calcium silicate to set under water, and also enough free lime to slake when water is added. Hydraulic limes are thus intermediate between natural cements and true limes. Little hydraulic lime is manufactured in the United States, but it is still much used in Europe.

NATURAL CEMENT.

Natural cement is formed by burning an impure limestone containing 15 to 40 per cent of silica, alumina, and iron oxides, at a temperature a little higher than that used in lime burning. Natural cement differs from hydraulic lime in that it does not contain sufficient free lime to cause slaking when water is added. During burning the carbon dioxide is driven off, and most of the lime combines with other constituents, forming silicates, aluminates, and ferrites of lime. As the burned mass will not slake, grinding is necessary. The ground material when wetted hardens or sets rapidly, both in air and under water. Unlike Portland cement, natural cement may vary greatly in composition. Also, it is burned at a much lower temperature than Portland cement, and its specific gravity is somewhat lower. The hydraulic property of natural cement does not depend on its magnesium content, but the presence of magnesium does not impair its value. Twenty-five years ago the cement produced in this country was chiefly natural cement, but for some years this type has formed only about 10 per cent of the total output.

PUZZOLAN CEMENT.

Puzzolan cement is made by mixing slaked lime with volcanic ash or blast-furnace slag. The mixture is finely ground but is not burned after mixing. When the powder is wetted it hardens like natural or Portland cement. It will set under water, and is claimed to give better service in water than in air.

PORTLAND CEMENT.

Portland cement is obtained by burning to the point of incipient fusion an intimate mixture of pulverized materials that contain calcium carbonate, silica, alumina, and iron oxide in proportions varying within certain narrow limits, and by pulverizing finely the clinker that results. The ground clinker sets under water, and is the strongest and most durable form of hydraulic cement.

HISTORICAL REVIEW OF CEMENT INDUSTRY.

ROMAN PUZZOLAN CEMENT.

Puzzolan cement is the most ancient form of hydraulic cement of which there is any record. The Romans manufactured it in the vicinity of Naples under the name of "Pozzuolana" and employed it in many of the early engineering works in Rome. The cement consisted of volcanic ash powdered and mixed with lime and possessed distinct hydraulic properties. In modern puzzolan cements blast-furnace slag is substituted for volcanic ash. Throughout the Middle Ages even the use of the primitive puzzolan cements seems to have been discontinued, lime mortar being used.

EUROPEAN NATURAL CEMENT.

The natural-cement industry had its beginning in an early discovery that certain limes would set under water. At first this property was thought to be due to the purity of the limestone from which the lime was made. About 1756 Smeaton, an English engineer, in investigating material suitable for constructing the Eddystone Lighthouse, discovered that the reverse was true, for the impure clayey limestones produced limes that would harden under water. The record of Smeaton's experiments[a] was not published until about 1791. His experiments were not carried to the point of making a true cement, but his conclusions regarding the effect of clay in limestone opened the way for further investigation.

In 1796, Parker[b] took out an English patent for a product which he termed "Roman cement," although it was much different from the cement used by the Romans. Parker's cement was made from concretions of clayey and limey material, burned at a temperature higher than that employed in burning lime. The burned product would not slake, but when ground to powder and wetted it would harden, in air and under water. A process for making natural cement of similar type was developed in France at about the same time as in England.

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[a] Eckel, E. C., and others, Portland cement materials and industry in the United States: U. S. Geol. Survey Bull. 522, 1913, p 19.

[b] Eckel, E. C., and others, work cited, p. 19.

EUROPEAN PORTLAND CEMENT.

In 1824, Joseph Aspdin,[a] of Leeds, England, received a British patent for a product which he named "Portland cement," because the set cement resembled the oolitic limestone of Portland, England. The specifications for the new cement were somewhat vague; a very pure limestone was to be burned to lime, the lime mixed with a definite quantity of clay, and the mixture pulverized wet. The wet mixture was to be dried and crushed and then calcined in a vertical kiln and finally the calcine was to be powdered. The patent does not state what proportions of lime and clay should be used, nor at what temperature the mixture should be burned.

The manufacture of Portland cement began in England and on the Continent shortly after the patent was issued. However, at that time natural cement was widely used, and as it could be manufactured more cheaply than the Portland cement, the growth of the latter industry was very slow.

THE CEMENT INDUSTRY IN THE UNITED STATES.

DEVELOPMENT AND USE OF NATURAL CEMENT.

In early days in the United States the greatest demand for cement was in canal construction, and consequently cement enterprises were developed along the courses of canals.

Natural cement was first used in America during the years 1818 and 1819 for the locks and walls of the middle section of the Erie Canal. Its use resulted from investigation of a peculiar type of lime, manufactured for this canal, that would not slake. One of the engineers, Canvass White, who had visited England and there become acquainted with natural cement, tested the material and found that it had hydraulic properties when ground and mixed with water. The material, which was really natural cement, was used in this canal and was much better than the lime that had been used theretofore. White took out a patent on this cement. According to Eckel,[b] the patent right for the State of New York was in 1825 purchased by that Commonwealth for $10,000, and the process rendered free to the citizens of the State.

This discovery was rapidly followed by the discovery of deposits of natural cement rock along the lines of other canals. Meade[c] states that in 1825 cement rock was discovered in Ulster County, N. Y., along the line of the Delaware & Hudson Canal, and in 1826 a mill was started at High Falls. In 1828 a mill was built in Ulster County, at Rosendale, which became a center of some importance. The cement made here was called "Rosendale cement," and this name is still applied to American natural cement. In 1829 natural cement was manufactured at Shippingport, near Louisville, Ky. In 1836, during the construction of the Chesapeake & Ohio Canal, natural cement was manufactured at Round Top, near Hancock, Md. Natural cement was manufactured for other canals as follows: The Illinois & Michigan Canal, at Utica, in 1838; the James River Canal, at Balcony Falls, Va., in 1848; the Lehigh Coal & Navigation Co. Canal, at Siegfried, Pa., in 1850.

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[a] Aspdin, Joseph, British patent 5022, issued Oct. 21, 1824.

[b] Eckel, E. C., and others, Portland cement materials and industry in the United States: U. S. Geol. Survey Bull. 522, 1913, p. 21.

[c] Meade, R. K., Portland cement, 2d ed., 1911, pp. 7–8.

THE PORTLAND CEMENT INDUSTRY.

According to Meade,[a] the first Portland cement made in the Lehigh district in the United States resulted from experiments begun by David O. Saylor in 1866 at Coplay, Pa., where natural cement had been manufactured for some years. After experimenting for some time with mixtures of ordinary cement rock and high-calcium cement rock, a mixture was found which, after being burned and ground, gave results equal to those obtained with imported English and German Portland cements. A feature of interest in connection with this achievement is the fact that the raw materials were very different from those employed in any European cement.

However, introducing the domestic product was difficult because European cements had an established record and American cement had no reputation. Eventually, through liberal advertising and guaranty, a market was established.

In 1872 a plant for making Portland cement from marl and clay was erected near Kalamazoo, Mich. In 1875 Portland cement made from limestone and clay was being manufactured at Wampum, Pa. Up to 1881 six plants were operated for the manufacture of Portland cement, three of which were failures and involved their promoters in heavy loss.

Before the Portland cement industry made any marked progress in the United States methods different from those employed in Europe were invented. Such modifications were necessary on account of the higher wages and cheaper fuel in the United States as contrasted with European countries. To meet these conditions it was necessary to cut down the labor cost for both burning and grinding by means of mechanical improvements. The stationary vertical kiln was, therefore, replaced by the rotary kiln, and the millstones formerly employed for grinding were replaced by modern grinding machinery, such as gyratory crushers and ball and tube mills. Soon after the rotary kilns were introduced it was found that the materials could be mixed and ground dry and fed directly to the kilns without wetting. This was the beginning of the dry process. In 1891, at Montezuma, N. Y., naturally wet raw materials (marl and clay) were charged into the kilns without preliminary drying. This was essentially the wet process as employed at present. Most of the cement manufactured in the United States is made by the dry process.

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[a] Meade, R. K., work cited, p. 10.

GROWTH OF THE INDUSTRY.

The Portland cement industry grew steadily from its beginning until 1908, and since that date its growth has been phenomenal. Few industries in the country can record such wonderful development as is indicated by an increase in production from approximately 3,500,000 barrels in 1898 to over 92,000,000 barrels in 1913. In the year 1900 the production of Portland cement in this country for the first time exceeded that of natural cement, and since then there has been a gradual decline in natural-cement manufacture. Chiefly because of overproduction and the immense stocks of Portland cement on hand, the production in 1914 and 1915 fell a little below that of 1913.[a]

The production of hydraulic cements in the United States from 1818 to 1915, as compiled by Burchard,[b] is shown in Table 1. A graphic illustration of the production of Portland and natural cements and the value of Portland cement for the period 1890 to 1915, prepared by Burchard,[c] is shown in figure 1.

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[a] Burchard, E. F., Cement: Mineral Resources of U. S. for 1915, U. S. Geol. Survey, 1916, pt. 2, p. 192.

[b] Burchard, E. F., work cited, p. 191.

[c] Preliminary estimates by the United States Geological Survey show that the output of Portland cement in 1917 amounted to 93,500,000 barrels, or about 1,100,000 barrels more than in 1916.

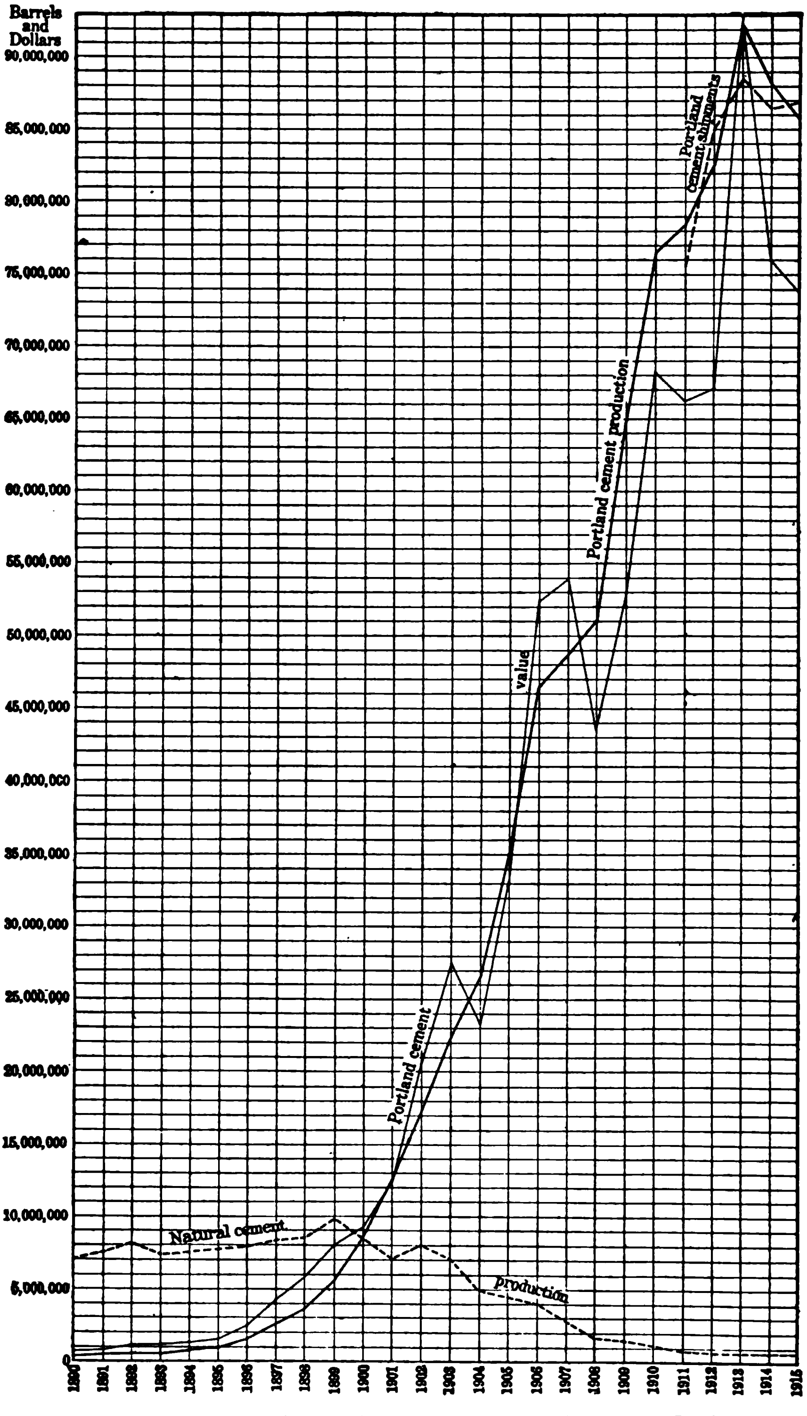

FIGURE 1.—Production of Portland and natural cements and value of Portland cement, 1890–1915, and shipments of Portland cement, 1911–1915. After Burchard.

TABLE 1.—*Total recorded production of principal hydraulic cements in the United States, 1818 to 1915, inclusive, in barrels.*

Year.	Natural cement.		Portland cement.		Puzzolan cement.		Total.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1818-1829	300,000	$246,000					300,000	$246,000
1830-1839	1,000,000	850,000					1,000,000	850,000
1840-1849	4,250,000	3,612,500					4,250,000	3,612,500
1850-1859	11,000,000	9,350,000					11,000,000	9,350,000
1860-1869	16,420,000	13,957,000					16,420,000	13,957,000
1870-1879	22,000,000	18,700,000	82,000	$246,000			22,082,000	18,946,000
1880	2,030,943	1,726,707	42,000	126,000			2,072,943	1,852,707
1881	2,440,000	2,379,000	60,000	150,000			2,500,000	2,529,000
1882	3,165,000	3,481,500	85,000	191,250			3,250,000	3,672,750
1883	4,000,000	4,100,000	90,000	193,500			4,090,000	4,293,500
1884	3,900,000	3,510,000	100,000	210,000			4,000,000	3,720,000
1885	4,000,000	3,200,000	150,000	292,500			4,150,000	3,492,500
1886	4,350,000	3,697,500	150,000	292,500			4,500,000	3,990,000
1887	6,692,744	5,186,877	250,000	487,500			6,942,744	5,674,377
1888	6,253,395	4,533,639	250,000	487,500			6,503,395	5,021,139
1889	6,531,876	4,702,951	300,000	500,000			6,831,876	5,202,951
1890	7,441,116	3,822,501	335,500	704,050			7,776,616	4,526,551
1891 [a]	7,767,979	3,671,147	454,813	967,467			8,222,792	4,638,614
1892	8,211,181	3,991,455	547,440	1,152,600			8,758,621	5,144,055
1893	7,411,815	3,251,757	590,652	1,158,138			8,002,467	4,409,895
1894	7,563,488	3,635,731	798,757	1,383,473			8,362,245	5,019,204
1895	7,741,077	3,895,424	990,324	1,586,830			8,731,401	5,482,254
1896	7,790,450	4,049,202	1,543,023	2,424,011	12,265	$12,250	9,345,738	6,485,463
1897	8,311,688	3,862,392	2,677,775	4,315,891	48,329	48,500	11,037,792	8,226,783
1898	8,418,924	3,888,728	3,692,284	5,970,773	233,000	198,050	12,344,208	10,057,551
1899	9,868,179	4,814,771	5,652,266	8,074,371	335,000	268,000	15,855,445	13,157,142
1900	8,383,519	3,728,848	8,482,020	9,280,525	365,611	274,208	17,231,150	13,283,581
1901	7,084,823	3,056,278	12,711,225	12,532,360	272,689	198,151	20,068,737	15,786,789
1902	8,044,305	4,076,630	17,230,644	20,864,078	478,555	425,672	25,753,504	25,366,380
1903	7,030,271	3,675,520	22,342,973	27,713,319	525,896	542,502	29,899,140	31,931,341
1904	4,866,331	2,450,150	26,505,881	23,355,119	303,045	226,651	31,675,257	26,031,920
1905	4,473,049	2,413,052	35,246,812	33,245,867	382,447	272,614	40,102,308	35,931,533
1906	4,055,797	2,423,170	46,463,424	52,466,186	481,224	412,921	51,000,445	55,302,277
1907	2,887,700	1,467,302	48,785,390	53,992,551	557,252	443,998	52,230,342	55,903,851
1908	1,686,862	834,509	51,072,612	43,547,679	151,451	95,468	52,910,925	44,477,656
1909	1,537,638	652,736	64,991,431	52,858,354	160,646	99,453	66,689,715	53,610,543
1910	1,139,239	483,006	76,549,951	68,205,800	95,951	63,286	77,785,141	68,752,092
1911	926,091	378,533	78,528,637	66,248,817	93,230	77,786	79,547,958	66,705,136
1912	821,231	367,222	82,438,096	67,016,928	91,864	77,363	83,351,191	67,461,513
1913	744,658	345,889	92,097,131	92,557,617	107,313	97,663	92,949,102	93,001,169
1914	751,285	351,370	88,230,170	81,789,368	68,311	63,358	89,049,766	82,204,096
1915	750,863	358,627	85,914,907	73,886,820	42,678	39,801	86,708,448	74,285,248
Total	234,323,417	149,179,644	856,433,138	810,475,954	4,806,757	3,937,695	1,095,563,312	963,593,293

[a] The figures for 1890 and previous years are estimates made at the close of each year and are believed to be substantially correct. Since 1890 the official figures are based on practically complete returns from all producers.

COMPOSITION OF PORTLAND CEMENT.

VARIATIONS IN COMPOSITION.

Portland cement contains approximately 60 to 70 per cent lime, 20 to 25 per cent silica, and 5 to 12 per cent alumina and iron oxide. The manner in which these various elements combine during the burning process has for many years been somewhat in doubt. Various investigators have studied the problem, and their conclusions do not agree. The chemical analysis of a Portland cement is somewhat difficult, but many reliable analyses have been made. Comparison of the analyses of various cements, all of good quality, shows considerable variation in composition. The major constituents may therefore vary within certain narrow limits without impairing the quality of the product. Basing his opinion on the results of many analyses, Meade[a] has pointed out that cements may vary in composition within the following limits and still conform with standard specifications:

Permissible limits of variation in composition of Portland cement.

Constituent.	Per cent.
Silica	19—25
Alumina	5— 9
Iron oxide	2— 4
Lime	60—64
Magnesia	1— 4
Sulphur trioxide	1— 1.75

According to specifications recently established,[b] the maximum permissible percentage of magnesia has been increased from 4 to 5, and that of sulphur trioxide from 1.75 to 2.

The fact that cements from different plants may vary several per cent in their contents of silica, lime, and alumina and yet give equally satisfactory results is difficult to explain. As the raw materials vary considerably in different localities, evidently the proportions of the various constituents are interdependent to a certain extent. The difficulty consists in determining what Meade terms the "proximate" composition; that is, the manner in which the various elements are combined, and their relation to each other.

INVESTIGATIONS OF "PROXIMATE" COMPOSITION.

As a result of investigations conducted by S. B. and W. B. Newberry[c] they concluded that Portland cement consisted of tricalcic silicate and dicalcic aluminate. They found that lime may be combined with silica in the proportion of 3 molecules to 1 and the product still have practically constant volume and good hardening properties, though hardening very slowly. With 3½ molecules of lime to 1 of silica the product is not sound and cracks in water. Also, they found that lime may be combined with alumina in the proportion of 2 molecules to 1, giving a product which sets quickly and shows constant volume and good hardening properties. With 2½ molecules of lime to 1 of alumina the product is not sound. The formula for tricalcic silicate, $3CaO.SiO_2$, corresponds to 2.8 parts of lime by weight to 1 part of silica. The formula for the dicalcic aluminate, $2CaO.Al_2O_3$, corresponds to 1.1 parts of lime by weight to 1 part of alumina. The permissible maximum percentage of lime is, then, the sum of the percentage of silica multiplied by 2.8 plus the percentage of alumina multiplied by 1.1.

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[a] Meade, R. K., Portland cement, 2d ed., 1911, p. 28.

[b] U. S. Government specifications for Portland cement: Bureau of Standards Circular 33, 3d ed., 1917, p. 27.

[c] Newberry, S. B., and Newberry, W. B., The constitution of hydraulic cements, 1897, p. 7.

The Newberrys deduced for Portland cement the general formula $X(3CaO.SiO_2)+Y(2CaO.Al_2O_3)$, in which X and Y are variable, depending on the relative proportions of silica and alumina present in the clay employed.

However, more recent investigations indicate that these conclusions were incorrect. Rankin and Wright,[a] after a long series of careful determinations, decided that Portland cement clinker made from the pure oxides consisted essentially of a mixture of tricalcic silicate, $3CaO.SiO_2$, dicalcic silicate, $2CaO.SiO_2$, and tricalcic aluminate, $3CaO.Al_2O_3$, with some of the substance $5CaO.3Al_2O_3$, and possibly a small amount of free lime. In a later publication Rankin[b] points out that the lime, and the substance $5CaO.Al_2O_3$, result from imperfect burning. In most cements they occur in small amounts.

A recent publication by the same author[c] describes a series of investigations conducted at the geophysical laboratory of the Carnegie Institute and the results of these studies, with a review of the various steps made by previous investigators leading toward a solution of the problem of the constitution of Portland cement. Rankin emphasizes the fact that the compound tricalcic silicate is the constituent of Portland cement that develops the greatest strength upon hardening. He concludes, therefore, that the essential process in the manufacture of cement is the formation of this compound, and that increase in the strength of the cement may be best brought about by increasing the content of tricalcic silicate. At present the average normal Portland cement contains only 30 to 35 per cent of this constituent. In order to work out a method of increasing the percentage of tricalcic silicate, further experimental work is needed. The rate of formation of this compound in a series of mixtures must be determined, and also its equilibrium relations at high temperatures.

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[a] Rankin, G. A., The ternary system $CaO-Al_2O_3-SiO_2$, with optical study by F. E. Wright: Am. Jour. Sci., vol. 39, January, 1915, p. 71.

[b] Rankin, G. A., The constituents of Portland cement clinker: Jour. Ind. and Eng. Chem., vol. 7, June, 1915, p. 473.

[c] Rankin, G. A., Portland cement: Jour. Franklin Inst., June, 1916, pp. 747–784.

In the past the manufacture of Portland cement received little scientific investigation, and the general influence of chemical composition on physical properties has been determined almost exclusively by practical experience. As noteworthy contributions to the problem of the constitution of Portland cement are now available, it seems likely that scientific research will play a more important part in the solution of practical cement problems. In fact, the way is now paved for laboratory investigations of practical methods for increasing the strength and the general quality of Portland cement.

OTHER INVESTIGATIONS.

Another field of investigation in which much remains to be done is a determination of the necessary modifications in the composition of cement to render it most suitable for certain climatic or other conditions affecting the structures in which it is employed.

The following examples illustrate problems of this nature. Cox[a] claims that a Portland cement best adapted for use in a tropical climate should have a high silica-alumina ratio, at least 3 parts of silicia to 1 part of alumina. Meade[b] refers to the claim made by numerous authorities that Portland cement containing high percentages of ferric oxide are best adapted for use in sea water, as such cements offer great resistance to the disintegrating action of the salt water. Blatchley[c] claims that the presence of sulphur hastens disintegration of Portland cement exposed to sea water.

RAW MATERIALS OF PORTLAND CEMENT.

ESSENTIAL ELEMENTS.

As pointed out on a previous page, Portland cement contains approximately 60 to 70 per cent lime, 20 to 25 per cent silica, and 5 to 12 per cent alumina and iron oxides. The essential elements are, therefore, calcium, silicon, aluminum, and iron, each of which is in nature usually combined in some way with oxygen. As these are the five chief elements of the earth's crust, it is evident that raw materials for cement manufacture are abundant and may be combined in innumerable ways to give the desired ratio.

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[a] Cox, A. J., Philippine raw cement materials: Phil. Jour. Sci., vol. 4, May, 1909, p. 217.

[b] Meade, R. K., Portland cement, 2d ed., 1911, p. 36.

[c] Blatchley, W. S., Portland cement: 25th Ann. Rept. Dept. Geol. and Nat. Res. Indiana, 1900, p. 15.

LIMITATIONS IN CHOICE OF RAW MATERIALS.

Although the raw materials for making cement are abundant and widely distributed, the necessity for maintaining a low manufacturing cost eliminates many sources of supply. Thus, silica is abundant in nature as the mineral quartz and alumina is found as corundum or emery, but these substances are so extremely hard and difficult to pulverize that they could not be used to advantage in making cement. Many silicates contain lime, iron, silica, or alumina, or combinations of these substances, but most of these also are difficult to quarry and costly to pulverize. Materials that may be employed for commercial manufacture with current prices of cement must, therefore, possess certain characteristics. The chemical composition must be such that a proper mixture may be readily procured and maintained. The physical characteristics must be such that quarrying, crushing, and pulverizing may be done at low cost, and the fusion points must be sufficiently low that clinker may be made without using an excessive amount of fuel.

SUITABLE MATERIALS.

Calcium in its most available state occurs as calcium carbonate in the form of limestone, marble, chalk, or marl. Vast quantities of alumina and silica occur in combination in the form of clay or related materials such as shale or slate. Limestone and clay are frequently mixed and grade into each other. When the clay content is 18 per cent or more the rock is termed argillaceous limestone or cement rock.

In addition to the rocks mentioned certain artificial by-products such as blast-furnace slag, ashes, oyster shells, or precipitated calcium carbonate from alkali works may be used as ingredients of Portland cement.

CLASSIFICATION OF MATERIALS.

The raw materials enumerated may be classified in two groups—the calcareous, supplying the major part of the lime, and the argillaceous, supplying the alumina and silica. Iron oxide may be present in any of the rocks mentioned and also in blast-furnace slag. Classification of the raw materials on this basis is as follows:

Classification of raw materials of Portland cement.

Calcareous.	Argillaceous.
Natural:	Natural:
Limestone.	Clay.
Marble.	Shale.
Chalk.	Slate.
Marl.	Cement rock.
Cement rock.	
Artificial:	Artificial:
Alkali waste.	Ashes.
Oyster shells.	Blast-furnace slag.

Cement rock consisting of both calcareous and argillaceous materials may fall in either group, its classification depending upon the predominating constituent. Most of the cement rock used for Portland cement in this country is argillaceous, requiring the addition of a certain amount of high-calcium rock. However, in a few localities the rock used is rather high in calcium and requires the addition of a small amount of clay or shale.

In ordinary cement mixtures, consisting of limestone or marl with clay or shale, the calcareous materials constitute about three-fourths of the mass, and argillaceous materials one-fourth. In cement rock the proportion may differ widely from this ratio.

COMMON IMPURITIES IN RAW MATERIALS.

SILICA.

In clay or shale silica is one of the essential constituents and can not be termed an impurity, but it may also occur in the limestone.

Silica when present in limestone may occur in three forms—(1) as free silica, in the form of vein quartz, sand, gravel, flint, or chert; (2) in combined form as clay, mica, hornblende, or other silicate mineral; (3) as layers or strata of hydrated silica, the result of chemical precipitation or organic action. The first form, free silica, is to be avoided in limestone. Quartz, flint, or chert are difficult to grind, and unless finely pulverized raise the fusion point of the mixture decidedly. The silicate form, clay, is usually desirable. If clay is present in proper amount for a cement, the mixture constitutes the ideal raw material. Other silicate minerals are to be avoided, some on the basis of undesirable constituents and others because of hardness or high fusibility. The third form, hydrated silica, is of rare occurrence.

IRON.

Iron may be present in oxide, sulphide, carbonate, or silicate forms. This metal, when present as oxide, carbonate, or silicate, is a useful flux, aiding in the combination of the lime and silica in the kiln. The iron sulphide, pyrite, in quantities exceeding 2 or 3 per cent is to be avoided. In any event the ferric oxide content of the finished cement should not exceed 4 per cent.

MAGNESIUM.

Magnesium is usually present in the limestone more than in the clay or shale and can scarcely be termed an impurity in limestone, being rather a constituent that replaces an equivalent amount of calcium. It has been claimed that any considerable proportion of magnesium causes Portland cement to expand and crack after a time. The effect of small proportions of magnesium has been the subject of many investigations, and of late years the opinion has been generally held that magnesium is less injurious than was formerly supposed. This opinion is reflected in the latest standard specifications for Portland cement[a] as prepared at a joint conference of representatives from the American Society of Civil Engineers, American Society of Testing Materials, and a United States Government departmental committee. According to these specifications the permissible percentage of magnesia is increased from 4 to 5.

SULPHUR.

Sulphur may be present as the mineral gypsum or calcium sulphate in the marl or clay, and as pyrite or iron sulphide in the limestone or in the coal used as fuel.

Blatchley[b] claims that sulphur is harmful in cement, especially when the concrete is exposed to sea water, as sulphur in any quantity hastens disintegration. Meade[c] asserts, however, that most of the sulphur in the raw material is burned in the kiln and therefore is not present in the finished cement. On this account it is probable that at least 5 or 6 per cent sulphur trioxide may be present in the raw material without impairing the quality of the cement.

ALKALIES.

According to Blatchley,[d] the Newberrys claimed that the alkalies, soda and potash, were probably of no value in promoting the combination of lime and silica, whereas the German chemist Schoch, on the other hand, claimed that they act as a flux and are of great benefit in the hardening of Portland cement.

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[a] U. S. Government specifications for Portland cement, Bureau of Standards Circular 33, 3d ed., 1917, p. 35.

[b] Blatchley, W. S., Portland cement: 25th Ann. Rept. Dept. Geol. and Nat. Res. Indiana, 1900, p. 15.

[c] Meade, R. K., work cited, p. 52.

[d] Blatchley, W. S., work cited, p. 15.

Meade[a] has shown by experiment that alkalies are useful fluxes.

The curtailment by the European war of the foreign supply of potash salts, and the consequent large advances in price, has led a number of cement manufacturers to add potash-recovery equipment to their plants. A brief and concise treatment of this subject is given by Porter.[b] Potash may, therefore, be regarded not only as a useful flux but also as an actual or potential source of a valuable by-product.

CALCAREOUS MATERIALS.

GENERAL SPECIFICATIONS FOR CALCAREOUS MATERIALS.

Calcareous materials to be suitable for cement manufacture should conform with the following conditions:

1. The rock should be free of concretions of iron minerals, should contain little free silica in the form of chert, flint, or quartz veins, and should be free of silicate minerals such as tremolite and diopside.
2. The silica and alumina contents should be sufficiently low and in such ratio that they will not interfere with the desired silica-alumina ratio in the finished product.
3. The rock should be sufficiently low in magnesium that the finished product will not contain more than 5 per cent magnesia.
4. The content of iron should be sufficiently low that the ferric oxide content in the finished cement does not exceed 4 per cent.
5. The sulphur content should be low.

LIMESTONE.

ORIGIN OF LIMESTONE.

Calcium is a common constituent of the rocks of the earth's crust. Calcium is always found in combination with other elements, as carbonate, sulphate, silicate, phosphate, or other forms. By the action of water and of acids at the surface of the earth or within the rocks, calcium is dissolved. Analyses of river waters show that they contain a considerable amount of carbonic acid. This aids in keeping the calcium in solution and it is carried by the rivers to the sea. Marine organisms of various types, such as foraminifera, corals, and mollusks, secrete shells consisting principally of calcium carbonate. As myriads of these organisms, in succeeding generations, live and die, their shells form vast accumulations at the bottom of the sea. Some of the limestones still show the fossils of which they were formed, but in others all trace of organic origin has been destroyed by the fine grinding to which the shells were subjected before their final consolidation. In addition to this mechanically precipitated material, calcium carbonate may be precipitated chemically from the sea water. Such mechanical and chemical precipitates, when consolidated, may form beds of limestone of great thickness. These may at later periods be elevated above the surface of the water by earth movements, and thus be rendered available for use.

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[a] Meade, R. K., work cited, p. 89.

[b] Porter, J. J.: Recovery of potash as by-product in manufacture of cement. Concrete, October, 1917, Cement Mill section, p. 27.

COMPOSITION OF LIMESTONE.

Limestone consists essentially of calcium carbonate, and when pure forms the mineral calcite. However, nearly all limestone contains certain other materials that may be classed as impurities. For a rock to be termed a limestone it must contain at least 50 per cent of calcium carbonate. The principal foreign elements in limestone are silica, iron oxide, alumina, magnesium carbonate, and the alkalies potash and soda. When 30 per cent or more of magnesium carbonate is present, the rock is termed a dolomite. When little silica, alumina, and iron oxide are present the relative proportion of each is of small moment to the cement manufacturer, but if the limestone contains a considerable proportion of these constituents, the relative amount of each should be such that when the shale or clay is added the resulting mixture shall contain silica and alumina in the proper ratio.

In determining the suitability of a limestone for cement making the composition of the shale or clay to be mixed with it must be considered, as the composition of the final mixture depends on the compositions of the original ingredients.

In general, pure limestones are much harder than argillaceous limestones, and consequently are less desirable for making cement. In South Dakota, Alabama, Mississippi, northern Texas, and southeastern Arkansas are certain rotton limestones or chalks that grind easily and are therefore desirable materials for making cement.

VARIETIES OF LIMESTONES.

Special names are applied to certain types of calcareous rocks differing from ordinary limestone in origin, texture, or composition.

Marble is limestone that, through the action of heat and pressure, has become more or less distinctly crystalline, although the term is also used in a commercial sense for any limestone that will take a good polish. Marl is a term applied to a loosely cemented mass of lime carbonate formed in a lake basin. Calcareous tufa and travertine are more or less compact limestones deposited around springs or along the courses of resulting streams. Oolitic limestone, so called because of its resemblance to fish roe, is made up of small rounded grains of lime carbonate having a concentrically laminated structure. Chalk is fine-grained limestone composed of particles of finely ground shells loosely cemented together. A siliceous or cherty limestone is one that contains considerable silica, an argillaceous limestone is one containing clay, and a carbonaceous or bituminous limestone is one containing carbonaceous matter.

PHYSICAL CHARACTERISTICS OF LIMESTONES.

Limestones vary greatly in such physical properties as hardness, color, weight, porosity, and texture. In color limestones range from pure white to black, depending on differences in chemical composition; in texture they may be amorphous, semicrystalline, or crystalline; and in compactness they vary from the loosely consolidated marls through the chalks to the compact, normal limestones and the still harder marbles. Porous limestone may weigh as low as 110 pounds per cubic foot, whereas the more compact varieties weigh 150 to 185 pounds to the foot.

From the standpoint of the cement manufacturer, porosity and hardness are the more important physical properties of a limestone. Porous limestones are capable of carrying a large content of water, the removal of which requires considerable fuel. The hardness has a direct bearing on the cost of quarrying, crushing, and grinding. Harder rocks pulverize with more difficulty and wear the grinding machinery much more rapidly than softer ones. These two properties, hardness and porosity, counteract each other to a great extent, for the harder limestones are usually of low porosity and vice versa.

The attitude of the beds and the presence or absence of open bedding planes or joints are structural features that have a direct bearing on the system of quarrying. As these chiefly affect drilling and blasting methods, they are considered in detail under "Drilling" and "Blasting" on pages 42 to 51.

CALCAREOUS MATERIALS USED IN CEMENT MANUFACTURE.

COMPACT LIMESTONE.

In the early days of Portland cement manufacture the amount of hard limestone used was small, argillaceous limestone, the so-called cement rock, being chiefly employed. When the process of making cement from hard limestone and shale or clay was placed on a profitable basis, this branch of the industry developed rapidly. Hard

The adaptability of the precipitated carbonate for use in cement manufacture depends largely on the nature of the limestone. If dolomitic limestone is used the resulting waste will be high in magnesium, but if high-calcium limestone is used the waste is of suitable character.

Alkali waste is employed to some extent as a Portland cement constituent, notably at Wyandotte, Mich.

OYSTER SHELLS.

It has been determined by analyses that ordinary commercial oyster shells have about the same composition as the purest limestones ordinarily quarried. The available supply of shells in certain coast and Gulf regions is sufficient to justify their use for cement manufacture. They are now employed for this purpose near Norfolk, Va., and Houston, Tex.

ARGILLACEOUS MATERIALS.

GENERAL SPECIFICATIONS FOR ARGILLACEOUS MATERIALS.

Argillaceous material for making cement should conform with the following specifications:

1. The material should be free from iron concretions and from silica in the form of flint and quartz.
2. The silica content should form 60 to 70 per cent of the mass.
3. The magnesium content should be sufficiently low to conform with standard specifications.
4. The presence of iron oxide is desirable as a flux, but the content of iron oxide in the finished product should not exceed 4 per cent.
5. For the manufacture of white cement, the ferric oxide content should be less than 0.5 per cent.
6. The sulphur content should be low.

CLAY.

DEFINITION OF CLAY.

Clay may be defined as an earthy substance consisting of a base of kaolin mixed with a greater or less proportion of mineral or organic impurities. Kaolin is a hydrous silicate of alumina, its composition being represented by the formula $Al_2O_3.2SiO_2.2H_2O$. Kaolin rarely occurs pure in nature, the more common impurities being silica sand, lime carbonate, iron oxide, and carbonaceous material.

ORIGIN OF CLAY.

Clays are all of sedimentary origin, and are derived from the disintegration and alteration of igneous, metamorphic, or sedimentary

rocks. Feldspar, pyroxene, and hornblende, which consist largely of silica and alumina, are the chief sources of kaolin that originates from igneous or metamorphic rocks. The more soluble elements are gradually leached out, leaving behind the less soluble silica and alumina.

In various places vast beds of clay have been formed by the disintegration of argillaceous limestones. Water containing carbon dioxide slowly dissolves and removes the calcium and magnesium carbonates, while the insoluble clay remains and accumulates.

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RESIDUAL CLAY.

Residual clay is formed through the disintegration of rocks in place, and is found replacing the parent rock and resting on unaltered masses of it. Residual clay results from the alteration of certain components of the rock mass, and those parts that have not altered to clay are usually present as impurities. Thus the residual clay from the disintegration of a granite may contain grains of unaltered quartz.

TRANSPORTED CLAY.

Most clays are not deposited in the localities where they originated, but have been transported by the agency of water or of ice. Alluvial clay is that which has been transported by streams, and deposited along their courses, on their flood planes or in the basins into which they flow.

Another type of transported clay is glacial clay. During the glacial period most of the northern part of America was covered with ice which in passing over the surface of the rocks eroded them. The loose sand, gravel, and clay were carried along by the ice and deposited at various distances from their original sources.

IMPURITIES IN CLAY.

As previously stated, pure kaolin is rare. Residual clays usually contain unaltered or partly altered grains of the parent rock. Transported clay may be freed of original impurities, but, on the other hand, impurities may be added in transit.

As clays are commonly formed through the disintegration of granites and related rocks, feldspar, mica, and quartz are common impurities. Feldspar acts as a flux and thus lowers the fusion point of a clay. Mica also acts as a flux, but fuses at a slightly higher temperature than feldspar. Quartz in any considerable amount is undesirable in clay to be employed for making cement.

The adaptability of the precipitated carbonate for use in cement manufacture depends largely on the nature of the limestone. If dolomitic limestone is used the resulting waste will be high in magnesium, but if high-calcium limestone is used the waste is of suitable character.

Alkali waste is employed to some extent as a Portland cement constituent, notably at Wyandotte, Mich.

OYSTER SHELLS.

It has been determined by analyses that ordinary commercial oyster shells have about the same composition as the purest limestones ordinarily quarried. The available supply of shells in certain coast and Gulf regions is sufficient to justify their use for cement manufacture. They are now employed for this purpose near Norfolk, Va., and Houston, Tex.

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limestone is the principal ingredient in practically 57 per cent of all the Portland cement manufactured in this country, and (see Table 2, p. 26) is by far the most widely used calcareous material in making cement. Hard limestone is usually of reasonably low porosity, but some types are difficult to pulverize.

CEMENT ROCK.

In certain parts of the United States, notably in the Lehigh River Valley of eastern Pennsylvania and western New Jersey, in the Shenandoah Valley of Virginia, and near Glens Falls, N. Y., are limestones having a high clay content. As such rocks contain all the chief constituents of Portland cement, they have been termed "cement rock." Cement rock was the chief source of raw material in the early days of the Portland cement industry in the United States and still constitutes an important source.

The cement rock quarried in the Lehigh district is mostly derived from the middle beds of the Trenton formation. In most places the beds dip 15° to 25°, usually northwest, but at a few quarries, particularly in New Jersey, the dip is much steeper. Most of the cement rock runs 60 to 70 per cent lime carbonate, hence the addition of a small amount of high-calcium limestone is necessary. The limestone may be obtained from beds in the lower part of the Trenton formation or from certain low-magnesium beds in the underlying Kittatinny formation. High-calcium limestone is shipped from outside points to a number of plants. Near Bath and Nazareth, Pa., the cement rock may carry 70 to 80 per cent of lime carbonate and the addition of limestone be unnecessary. In some places, on the other hand, the rock is so high in lime that the addition of a small amount of clay or shale is necessary. A high-calcium cement rock, requiring a small addition of clay, is usually more desirable than one requiring the addition of limestone, because the cement rock is usually overlain with clay, which is readily available, whereas the limestone may have to be shipped in.

In a few places the cement rock approximates so nearly the desired proportions for a cement mixture that by careful and judicious quarrying no limestone or shale is necessary. Such a rock, properly proportioned in nature, constitutes the ideal raw material for cement.

Cement rock is commonly intersected by close bedding seams and joint planes, consequently when great masses are hurled down in blasting they readily break into small pieces. Not only does cement rock quarry easily, as a rule, but it offers certain advantages in milling. As the particles of clay and limestone are intimately mixed by nature, the rock does not need to be so finely ground as limestone must be for intimate mixing with clay or shale. Moreover, cement rock is, as a rule, much easier to grind and pulverize than hard limestone. Such rock may, therefore, be regarded as one of the most desirable raw materials for making cement.

MARBLE.

Marble is an extremely compact or crystalline form of limestone. It is usually much harder than ordinary limestone, and consequently is somewhat difficult to quarry and pulverize. Marble is not commonly used as a cement material. It is employed by a company operating near Union Bridge, Md.

CHALK.

Chalk, being much softer than limestone and easily excavated and pulverized, is a desirable raw material for cement manufacture. The chief disadvantage of chalk is its high porosity, which permits excessive absorption of water during rainy seasons. Where chalk is employed large covered bins in which a supply can be stored for use during wet seasons is desirable.

MARL.

Marl is soft, unconsolidated lime carbonate that has been deposited in the bottoms of existing or extinct lakes. The lime carbonate is precipitated from solution in the lake water by the agency of certain algæ or water plants. The chief impurities in marl are clay, organic matter, and magnesium carbonate. The clay content, if of suitable composition and limited in amount, is not regarded as undesirable. The organic matter is burned out in the kilns and has little or no effect on the finished product.

As most marl deposits are partly or completely submerged, the marl is won by dredging. The wet material may be hauled to the plant in cars, or may be pumped as a slurry. The process of winning marl is relatively cheap, but this advantage is largely counteracted by the cost of fuel for driving off the water.

ALKALI WASTE.

Limestone is employed in the manufacture of caustic soda from common salt. The calcium is precipitated in carbonate form and removed as waste. Where the Leblanc process is employed the waste contains too large a proportion of sulphides to be used as an ingredient of cement. The ammonia process, however, yields lime carbonate relatively free from sulphur.

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Most clay contains iron. The metal may be present as the oxide in such minerals as magnetite, hematite, and limonite, or as the sulphide in the mineral pyrite.

Calcium and magnesium carbonates are common constituents of clay. As pointed out on a previous page, a high proportion of calcium carbonate may be present, forming a cement rock. Magnesium carbonate is undesirable except in small amounts.

Gypsum or calcium sulphate is present in many clays. Its presence may increase the amount of sulphur in the raw mix beyond the permissible percentage.

The alkalies, potash and soda, are commonly present. They act as fluxes, tending to lower the fusion point of the clay.

COMPOSITION OF CLAY FOR CEMENT MANUFACTURE.

In judging the suitability of a clay for making cement, the condition of the free silica should be noted with care. Practically all clay contains some uncombined silica in the form of quartz sand or pebbles. The proportion of free silica should be low, and it is desirable that the sand, if present, should be in a finely divided condition. Meade[a] claims that clay containing more than 5 per cent of sand in grains that will not pass through a 100-mesh sieve, is unsuitable for cement manufacture.

The ratio between the silica, alumina, and iron contents is important. The ratio of silica to alumina should range between 3 to 1 and 4 to 1. The clay should not contain more iron oxide than alumina, the best proportion being about 1 of iron oxide to 3 of alumina. The sum of the alumina and the iron oxide contents should not be more than one-half the silica content, and the nearer it approaches one-third, the better. If the percentage of alumina is too high, the cement sets too fast, and also is weak. Plants have been known to suffer heavy financial loss through the rejection of inferior cement on account of the clay being low in silica. This difficulty could be easily overcome by adding free silica in some form to the mixture.

The sum of the silica and alumina percentages in hydrous clay should not exceed 87 per cent, thus leaving at least 13 per cent for combined water and fluxes like ferric oxide, potash, and soda. The presence of these fluxes, especially ferric oxide, promotes the formation of an easily fusible magma. The lower the fusing point of the magma the more readily will the cement mixture sinter or clinker, and the lower will be the cost of burning. However, the content of alkalies should not be more than 3 per cent, as an excess of these is likely to cause unsoundness and quick setting.

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[a] Meade, R. K., Portland cement, 2d ed., 1911, p. 54.

SHALE AND SLATE.

ORIGIN.

Shale consists of mud or clay that has been deposited in water and subsequently through pressure been consolidated into rock. The pressure may be due either to the weight of overlying rock formed by subsequent deposition, to forces to which the rocks were subjected during mountain-forming periods, or to both of these causes combined. Under further pressure and increased temperature the shale may be changed in character and structure, chemical changes may take place, recrystallization be brought about, and a perfect cleavage or fissility developed. The resulting rock is termed slate.

COMPOSITION.

As shale is merely solidified clay, shale to be used for cement manufacture should conform with all the conditions prescribed for clay. Slate has usually undergone some recrystallization, but its chemical composition is essentially that of the original clay from which it was formed. Hence, all that has been stated relative to impurities and relative proportions of clay constituents is equally applicable to shale and slate.

PHYSICAL CHARACTERISTICS.

Clay is usually earthy, but shale and slate are more firmly consolidated and more closely resemble hard rocks such as limestone or cement rock. This difference in physical character is of some importance in the selection of suitable material for cement. Clay is the most suitable argillaceous material to mix with marl, as the two materials are of similar physical character. If clay is mixed with coarsely ground limestone the two materials tend to segregate; that is, if the mixture falls in a stream onto a conical heap below, as in filling a storage bin through a spout, the finely divided clay tends to remain in the center of the pile and the limestone fragments to roll to the sides. This separation of materials tends to cause lack of uniformity when the mixture is drawn from the bin. If shale or slate is mixed with limestone the constituents are more nearly alike in character and such segregation is not so likely to occur. It is better, therefore, to mix substances of like physical characteristics—shale or slate with limestone or cement rock, and clay with marl.

Where limestone deposits are close to slate quarries, waste slate might be utilized as a cement constituent. The utilization of the waste would thus be beneficial to both industries.

BLAST-FURNACE SLAG.

True Portland cement may be made from a mixture of ground blast-furnace slag and limestone burned together. Slag consists essentially of lime, silica, and alumina, with small contents of iron oxide, magnesium, and sulphur. As an ingredient for cement slag may, therefore, be regarded as very impure limestone, or as very calcareous clay from which the carbon dioxide has been driven off.

The suitability of a blast-furnace slag as a cement constituent depends largely on the nature of the limestone used for flux in the blast furnace. A magnesium limestone or dolomite naturally produces a slag high in magnesium. Hence only slags from furnaces that employ high-calcium limestone for flux can be used for making Portland cement.

ASHES.

Coal ashes are used to some extent to replace or supplement clay or shale in cement mixtures. It is claimed by one company that ashes tend to decrease the rate of setting in the finished cement.

COMBINATIONS OF RAW MATERIALS.

In a few localities cement rock has been found that contains lime, silica, alumina, and iron oxide in almost the proper proportions for a cement mixture. However, such rock is rare, and as a rule a mixture of calcareous and argillaceous materials is necessary. The various combinations of raw materials employed in cement plants in the United States are outlined in the following paragraphs.

CEMENT ROCK AND LIMESTONE.

Most of the cement rock now used in the Lehigh district is so low in calcium as to require the addition of a small amount of high-calcium limestone. In some quarries suitable limestone is found underlying the cement rock, whereas for others it must be shipped in from outside points. Cement rock and limestone are also available in Virginia and in some of the Pacific Coast States, but are not used extensively.

CEMENT ROCK AND SHALE.

In a few quarries in the Lehigh district and in one quarry near Glens Falls, N. Y., the cement rock contains an excess of calcium, and the addition of a small amount of shale is necessary.

HARD LIMESTONE AND CLAY OR SHALE.

As stated on page 18, hard limestone is the chief calcareous constituent of Portland cement made in this country. The combination of hard limestone with clay or shale is practiced by cement plants in many States and in widely separated localities. The substitution of ashes for part of the shale has been mentioned.

CHALKY LIMESTONE AND CLAY.

Chalky limestone is easier to quarry than hard limestone, and carries less water than marl. It is available for cement manufacture in three localities: (1) Parts of Texas, Oklahoma, and Arkansas; (2) parts of Alabama and Mississippi; and (3) parts of North Dakota and South Dakota.

MARL AND CLAY.

Marl and clay are used for making cement in Ohio, Michigan, Indiana, and central New York.

ALKALI WASTE AND CLAY.

Precipitated calcium carbonate from alkali works is a limited source of supply as a cement constituent and is now utilized only at Wyandotte, Mich.

SLAG AND LIMESTONE.

Blast-furnace slag and limestone are used for the manufacture of cement in Illinois, Ohio, and Pennsylvania.

OYSTER SHELLS AND CLAY.

Oyster shells, which have about the same composition as limestone, are (see p. 20) employed in a few localities where the supply of shells is adequate.

RELATIVE IMPORTANCE OF MATERIALS USED.

Table 2, compiled by Burchard,[a] indicates the relative importance of the chief sources of raw materials for the period 1898 to 1914, inclusive.

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[a] Burchard, E. F., Cement: Mineral Resources of U. S. for 1914, pt. 2; U. S. Geol. Survey, 1915, p. 232.

TABLE 2.—*Production, in barrels, and percentage of total output of Portland cement in the United States according to type of material used, 1898–1914.*

Year.	Type 1. Cement rock and pure limestone.		Type 2. Limestone and clay or shale.		Type 3. Marl and clay.		Type 4. Blast-furnace slag and limestone.	
	Quantity.	Percentage.	Quantity.	Percentage.	Quantity.	Percentage.	Quantity.	Percentage.
1898	2,764,694	74.9	365,408	9.9	562,092	15.2	....	....
1899	4,010,132	70.9	546,200	9.7	1,095,934	19.4	....	....
1900	5,960,739	70.3	1,034,041	12.2	1,454,797	17.1	32,443	0.4
1901	8,503,500	66.9	2,042,209	16.1	2,001,200	15.7	164,316	1.3
1902	10,953,178	63.6	3,738,303	21.7	2,220,453	12.9	318,710	1.8
1903	12,493,694	55.9	6,333,403	28.3	3,052,946	13.7	462,930	2.1
1904	15,173,391	57.2	7,526,323	28.4	3,332,873	12.6	473,294	1.8
1905	18,454,902	52.4	11,172,389	31.7	3,884,178	11.0	1,735,343	4.9
1906	23,896,951	51.4	16,532,212	35.6	3,958,201	8.5	2,076,000	4.5
1907	25,859,095	53.0	17,190,697	35.2	3,606,598	7.4	2,129,000	4.4
1908	20,678,693	40.6	23,047,707	45.0	2,811,212	5.5	4,535,300	8.9
1909	24,274,047	37.3	32,219,365	49.6	2,711,219	4.2	5,786,800	8.9
1910	26,520,911	34.6	39,720,320	51.9	3,307,220	4.3	7,001,500	9.2
1911	26,812,129	34.1	40,665,332	51.8	3,314,176	4.2	7,737,000	9.9
1912	24,712,780	30.0	44,607,776	54.1	2,467,368	3.0	10,650,172	12.9
1913	29,333,490	31.8	47,831,863	51.9	3,734,778	4.1	11,197,000	12.2
1914	24,907,047	28.2	50,168,813	56.9	4,038,310	4.6	9,116,000	10.3

PROPORTIONING THE MIXTURE.

LIMITATIONS OF DISCUSSION.

Meade [a] has discussed the proportioning of raw materials at length, and readers particularly interested in the subject may consult the publication cited, or others of similar character. In a publication like the present one, dealing primarily with the process of winning the raw materials from the earth, the principles and the method of obtaining a suitable proportion of raw materials can be dealt with only in a general way.

NEED OF DEFINITE PROPORTIONS.

In discussing the composition of Portland cement it has been pointed out that the properties of cements may be materially changed by very small variations in the proportions of the essential elements. For example, cements high in lime or silica are slow-setting, whereas those high in alumina are quick-setting. Variations beyond certain narrow limits may cause the cement to be of poor quality. The cement manufacturer must, therefore, control his mixture by certain definite rules.

FORMULAS FOR CALCULATING MIXTURE.

Certain general formulas are applicable to most cement materials, though peculiarities in composition may require minor modifications.

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[a] Meade, R. K., Portland cement, 2d ed., 1911, pp. 69–93.

The Newberrys[a] based their formula on the presumption that Portland cement consisted of tricalcic silicate and dicalcic aluminate. Tricalcic silicate ($3CaO.SiO_2$) consists of 2.8 parts of lime by weight to 1 part of silica; and dicalcic aluminate ($2CaO.SiO_2$), 1.1 parts of lime by weight to 1 part of alumina. The maximum permissible percentage of lime, then, is equal to the sum of the percentage of silica multiplied by 2.8 plus the percentage of alumina multiplied by 1.1.

The following rule was deduced: Multiply the percentage of silica by 2.8, and the percentage of alumina by 1.1, add the products, and the sum will be the number of parts of lime required for 100 parts of clay.

As 2.8 parts of lime correspond to 5 parts of lime carbonate, and 1.1 parts lime correspond to 2 parts of lime carbonate, it follows that five times the percentage of silica plus twice the percentage of alumina equals the number of parts of carbonate of lime required for 100 parts of clay.

It has been established that the ratio between the percentage of lime and the combined percentages of silica, alumina, and iron should be within the limits 1 to 1.8 and 1 to 2.2. Many chemists assume a fixed ratio of 2, though others obtain better results by slightly increasing or decreasing this ratio.

After the raw clay and limestone have been analyzed the proportions of each necessary for any given ratio may be determined as follows:

Let—

M =the ratio of lime to silica, alumina, and iron;
S =the oxides of silica, alumina, and iron in the clay;
S_1=the oxides of silica, alumina, and iron in the limestone;
C =the calcium oxide in the limestone;
C_1=the calcium oxide in the clay.

Then—

$$\frac{\text{Limestone}}{\text{Clay}}=\frac{MS-C_1}{C-MS_1}.$$

The chief defect of this formula is that the relative proportions of silica, alumina, and iron oxide are not taken into consideration, but for dealing with well-balanced raw materials it is satisfactory. More accurate methods for calculating these proportions are given by various authors.

CALCULATION ON BASIS OF FIXED LIME STANDARD.

A formula such as the one given is used to determine the best mixture of any given raw materials and to check the mixture from time to time. However, the "fixed lime standard" is found to be more practical in actual millwork. After experience has shown that a certain percentage of lime carbonate gives satisfactory results, the operator maintains this lime standard as nearly as possible, on the assumption that the ratio of other constituents is approximately self-adjusting. The method is satisfactory where the raw materials are of fairly constant composition, but if marked fluctuations in composition are likely to occur the mixture should be checked frequently by the application of a more accurate formula.

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[a] Newberry, S. B., and Newberry, W. B., The constitution of hydraulic cements, 1897, p. 7.

In the Lehigh district the percentage of carbonate of lime in the raw mixture varies from 74.5 to 75.5, and as the limestone and cement rock are of fairly constant composition, the fixed lime standard is in common use.

Meade[a] gives the following simple formulas for determining the proper mixture on this basis. The first formula is for determining the percentage of limestone to be added to a given cement rock or clay to make a given mixture.

Let—

X=percentage of limestone necessary,
L=percentage of $CaCO_3$ in limestone,
R=percentage of $CaCO_3$ in cement rock or clay,
M=percentage of $CaCO_3$ desired in mixture,

Then—

$$X=\frac{M-R}{L-M}\times 100$$

The second formula is for determining the percentage of shale or clay to be added to a given marl or limestone to make a given mixture, and is as follows:

Let—

X=percentage of clay or shale necessary,
C=percentage of CaO in clay or shale,
L=percentage of CaO in marl or limestone,
M=percentage of CaO in desired mixture.

Then—

$$X=\frac{L-M}{M-C}\times 100$$

FLUXES.

The addition of certain substances to promote combination of silica and lime in the kiln is necessary in certain plants working on poor raw materials, such as cherty limestone or high-silica and low-alumina clay. The common fluxes used in cement making are iron oxide, fluorspar, cryolite, and certain alkaline compounds. Where fairly good raw materials are available, the use of fluxes usually should be avoided, as the disadvantages may more than outweigh the advantages.

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[a] Meade, R. K., work cited, p. 80.

PROCESSES OF MANUFACTURE.

NATURAL CEMENT.

The cement rock used for the manufacture of natural cement requires no pulverizing or mixing prior to burning. It is burned in vertical kilns similar to those employed in lime burning, though usually larger. The kilns are operated continuously, the rock and fuel being fed in at the top, mixed together or in alternating layers. The stone is burned at a temperature a little higher than that employed in lime burning. The burned stone, or clinker, is drawn from the bottoms of the kilns, cooled, and ground to powder.

PORTLAND CEMENT.

During the early period of the Portland cement industry all European and American plants used the wet process only. As stationary kilns were used the material had to be wetted and molded into bricks to give a free draft in the kiln.

With the introduction of the rotary kiln the dry process was made practicable. The dry process involves, first, the crushing and drying of the limestone or cement rock and the addition of the necessary amount of clay, shale, or limestone, whichever is used, these being also crushed and dried. The mixture is then finely pulverized. With cement rock satisfactory results are obtained if the material is sufficiently pulverized that 85 per cent passes through a 100-mesh sieve. If limestone and shale are used, finer grinding is necessary; for the best results, 95 to 98 per cent should pass through a 100-mesh sieve. The properly proportioned and finely pulverized mixture is then fed into the kilns.

In the wet process the materials are mixed and usually pulverized in a wet condition. The mixture known as "slurry," having the consistency of thin mud, is fed by pumps or by screw conveyors into tanks, where it is throughly mixed either by mechanical or air agitation. The mixture enters the kilns as slurry, the water being driven off by evaporation in the upper part of the kilns.

Modern rotary kilns are 5 to 9 feet in diameter and 60 to 200 feet long, the longer kilns representing a more modern development. They are cylindrical in form and are made of sheet steel lined with fire brick. They slope gently downward toward the firing end. Fuel consisting of powdered coal, gas, or oil is blown in through a small pipe at the lower end of the kiln. The cement mixture is fed in at the upper end, and as the kiln rotates slowly the mixture works its way toward the lower end, where it is discharged as clinker burned to the point of incipient vitrification.

The clinker is cooled and seasoned for some time before grinding. Two to 3 per cent of retarder, usually gypsum, is added to the clinker to prevent too rapid setting of the cement, and the mixture is pulverized and conveyed to the stock house. To prepare for shipment, the cement is packed in barrels of 380 pounds, or bags of 95 pounds, each. Automatic packing devices are in common use.

PROSPECTING.

NEED OF THOROUGH PROSPECTING.

It is extremely unwise to begin quarrying rock for cement manufacture without first driving enough prospect holes to determine whether a sufficient supply of material is available. The opening of a new quarry involves considerable expense for equipment and labor, and should, therefore, not be undertaken without definite assurance of practical operation.

CHEMICAL COMPOSITION OF RAW MATERIALS.

The matter of supreme importance to the cement manufacturer is the chemical composition of the rock. In this respect there is a wide difference between prospecting for rock to be used in making cement and for rock to be used for structural purposes. When rock is to be quarried for building purposes, the soundness, color, texture, and condition of cementation are of more consequence than the chemical composition. On the other hand, the physical properties of rock quarried for cement manufacture, although they may have considerable influence on the method of blasting, are secondary in importance to chemical composition.

In prospecting to determine the suitability of a deposit of clay, limestone, or cement rock, for cement manufacture the deposit should be sampled carefully and systematically in order that the samples may fairly represent the entire deposit. The less homogeneous the deposit is, the more samples should be taken and the more carefully should the work be done. Neglect to make a proper preliminary examination of a deposit of either clay or limestone before beginning quarrying may result in the loss of thousands of dollars. The substances that must be most carefully avoided are magnesium and free silica, although small percentages of either may usually be safely allowed.

INTERRELATION OF VARIOUS CONSTITUENTS.

In determining the suitability of raw cement materials the composition of the limestone or marl and that of the clay or shale must not be considered alone, for each must be considered in its relation to the other. Raw cement material usually consists of a mixture of two substances, and a deficiency in one may be compensated by a corresponding excess in the other. Thus a low silica shale may be satisfactory when mixed with a siliceous limestone.

In limestone the proportion of silica and alumina to calcium may vary widely and still the rock may be satisfactory. Thus limestone having as high as 98 or 99 per cent calcium carbonate may be satisfactory if the necessary shale or clay to mix with it is readily available. On the other hand, the rock may be very low in calcium—it may even contain less calcium than is necessary for a cement mixture—and still be satisfactory if high-calcium limestone to mix with it is available. Hence considerable latitude may be permitted in the composition of the main rock mass, the limits of variation being governed by the availability of the raw material that must be added in order to give the proper mix.

METHODS OF PROSPECTING FOR CLAY.

Bleininger[a] suggests that in sampling clay, if samples taken from an outcrop show a satisfactory composition, the surface should be removed and samples taken from within the bank. If further outcrops of the same deposit can be found they should all be sampled. The area comprising the clay deposit should then be surveyed and divided into squares whose sides should range between 50 and 100 feet in length, depending on the thickness of the deposit. If the deposit is thin and covers a wide area the squares should be larger than where it is thick. At the center of every square a test hole should be drilled and the material brought up by the drill put aside for analysis.

If the clay is reasonably soft and free from stones, the drilling may be done with a soil auger. For harder material a simple type of churn drill, operated either by hand or by power, may be employed.

EXAMPLES OF INSUFFICIENT PROSPECTING.

As an example of loss due to insufficient prospecting, attention may be directed to a certain Pennsylvania quarry where a depth of 10 to 15 feet of overburden was removed from a considerable area of rock, and it was subsequently found that the rock was useless for the manufacture of cement. The drilling of a single hole would have revealed the inferior nature of the rock and thus saved the expense of useless stripping.

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[a] Bleininger, A. V., The manufacture of hydraulic cements: Geol. Survey of Ohio, ser. 4, Bull. 3, 1904, p. 103.

AVAILABILITY OF ROCK LEDGE.

Next in importance to the composition of the rock is its availability for quarrying. The thickness and character of the overburden, and the effect of continued development on the depth of overburden, should be carefully ascertained. The relation of the thickness of available stone to depth of overburden is important, for thin beds would necessitate wide lateral development and, therefore, the removal of relatively much more stripping than where deep quarrying may be employed.

If limestone and shale quarries are not close together, it is desirable to build the mill near the limestone quarry, as about three or four parts of limestone are used to one part of shale.

UTILIZATION OF GRAVITY SYSTEMS.

The quarry and the cement plant should, if possible, be so situated that the raw materials can be conveyed to the plant and pass through each process to the final bagging and packing by means of a gravity system. If the materials must be raised repeatedly to higher levels, much power is required, whereas if the materials can be moved by gravity the cost of production may be greatly reduced.

Quarries should, wherever possible, be so located that their floors lie above drainage level.

QUARRY METHODS AND EQUIPMENT.

GENERAL METHOD.

The raw materials for the manufacture of cement are excavated by three different methods, quarrying, mining, and dredging.

Open-pit quarrying is the method most commonly employed. It is claimed that about 85 per cent of all the raw material used in the manufacture of cement is quarried. Mining of cement material is a modification used where the overburden is so thick that stripping would not be practicable. Dredging is employed in the removal of marl from lake bottoms.

FACTORS GOVERNING PLAN OF QUARRYING.

The plan of quarry development that will give the best results is governed by the depth of stripping, the thickness and attitude of the beds, the structure of the rock, and the uniformity of its chemical composition in the same and in successive beds.

If the overburden is thick, deep quarrying should be pursued if conditions will permit, for thus the maximum quantity of rock is obtained for a minimum of stripped surface. If the good rock is not thick enough to make deep excavation possible, cheap methods of stripping must be employed to make quarrying profitable.

Where the overburden is thin and a great thickness of rock is available, the depth to which the quarry may be worked is a matter of choice. The quarryman should seek to develop a quarry face of the height that can be worked the most cheaply. Usually a face 30 feet high or higher is more cheaply quarried than one less than 30 feet in height.

If the beds are flat the direction of the quarry walls is influenced only by surface contour, by structural features such as joint planes, or by the most convenient situation for transportation lines. However, for steeply inclined beds it is usually better to make the quarry face at right angles to the strike of the beds and thus cut across successive strata. When the face is in this direction the rock is more easily blasted and a more uniform mixture is obtained.

Every quarry plan is influenced more or less by drainage conditions. If a large flow of water is encountered, automatic drainage at a high level, leaving sufficient rock face above water level, would discourage deep quarrying which involved excessive pumping.

In beds dipping at a low angle it is usually wise to quarry in such a manner that the face is moved up the dip. By this method drainage from the face is insured, the rock is thrown down and loaded more easily than when worked in the opposite direction, and the loaded cars may be partly or entirely moved by gravity.

In choosing a suitable level for a bench or quarry floor in a rock deposit where the beds are approximately horizontal, the operator should be governed chiefly by the position of open bedding planes. Such planes promote effective blasting and also make a smooth floor, which greatly simplifies the laying of tracks and the loading of rock.

Where the strata are extremely folded and contorted with synclines, anticlines, and faults the process of quarrying may be more complicated. Especially is this the case where dolomite and limestone are interbedded as in a certain Georgia quarry. In that quarry the beds are folded, have irregular axes, and the structure is complicated by faults. The dolomite, in general, forms the axes of the anticlines and is interbedded with the limestone at various levels. It is a difficult matter to quarry this rock in such a way as to avoid the dolomite. The method used is, in general, to maintain a quarry face at right angles to the axes of the folds and to work out the synclines. The dolomite, however, occurs in many unexpected places, and the problem is therefore beset with many uncertainties. The writer is confident that working out of the structural relations of the dolomite and the limestone by a competent geologist, and careful mapping of the quarry area would be of immense value to the company. Cross sections showing the relation of the dolomite to the limestone and the position and throw of faults would be of especial value. At many quarries geologic study and the careful preparation of maps would probably be of great help in laying out and developing the quarry. In such investigations emphasis should be placed on the relation of good rock to impure rock. As far as practicable the position of each in the deposit should be ascertained in order that a plan of development may be worked out by which the good rock may be obtained and the defective material avoided.

POWER.

Quarry drills are commonly operated by compressed air, steam, or electricity obtained from the central power house of the cement plant. Where the quarry is situated at a considerable distance from the cement plant a separate power plant may be necessary. Shovels for loading rock or overburden may be operated by electricity, compressed air, or steam. The steam shovel is the more common type, and coal is almost invariably used for fuel. The power cost depends largely on the availability of a coal supply. In one of the Ohio quarries visited, a thin bed of coal beneath the limestone supplied fuel for the steam shovels.

In quarries of wide extent, situated in localities where the winters are severe, the loss of power in long steam lines may be considerable. If it is found advisable to use steam for drills in such quarries, power losses may be minimized by establishing several small boiler houses near the drilling operations rather than conveying the steam long distances from a central generating plant.

A detailed discussion of power plants is not within the scope of this bulletin. Much useful information may be obtained by consulting the work of Brunton and Davis.[a]

STRIPPING.

NATURE AND EXTENT OF OVERBURDEN.

Stripping or removal of overburden is a very variable factor in quarrying. Some deposits are exposed and no stripping is necessary. In other places the removal of 40 to 50 feet of cover may be necessary, whereas a depth in excess of 50 feet is common. As a rule no attempt is made to remove such excessive overburden. The rock deposit is either classed as unavailable or tunnel methods of quarrying may be employed.

The overburden may consist of clay, sand, gravel, or a mixture of these substances, and may contain bowlders of various sizes. It may be soft and easily excavated by hand or with a steam shovel or it may be so hard that blasting is necessary. All these factors must be considered in determining the best means of removal.

OVERBURDEN QUARRIED WITH ROCK.

The chemical composition of the overburden is of considerable importance. If the underlying rock is a high-calcium limestone requiring the addition of clay or shale, it is possible that the overburden may be used as a constituent of the raw mix. Under such conditions stripping may not be necessary. Instances have been observed where a depth of 2 to 4 feet of clay or weathered shale is not removed from the rock surface, but is shot down and loaded with the rock. There are two objections to this process which condemn it in the minds of some operators. First, in wet weather the clay makes the rock muddy and difficult to handle and crush, particularly where gyratory crushers are employed. In the second place, the presence of large quantities of clay, which are often imperfectly mixed with the rock, makes it difficult for the chemist to maintain a uniform mixture.

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[a] Brunton, D. W., and Davis, J. A., Safety and efficiency in mine tunneling: Bull. 57, Bureau of Mines, 1914, pp. 46–78.

It is doubtful whether shooting down a thickness of 2 to 4 feet of clay with the rock is advisable. However, the fact that clay is one of the constituents commonly added to limestone to make cement obviates the necessity for clean stripping in many places, and thus relieves the quarryman of an operation which is usually tedious and costly.

Where the depth of overburden varies in different parts of the quarry, the feasibility of shooting down the overburden with the rock depends to some extent on the method of quarrying. In one Kansas quarry the overburden varies in thickness in different parts of the quarry from a few inches to 10 feet. At one time the overburden was shot down with the rock. Hand-loading methods were then employed, and while some cars were loaded with an excessive amount of earth, others from that part of the quarry where the overburden was thin were loaded almost entirely with rock. By sending the cars to the crusher in proper order a uniform mix was maintained. Later a steam shovel was introduced for loading rock in the quarry, hence the rock could be taken only from one part of the quarry at one time. If the overburden at that place was thick the proportion of clay loaded with the rock would be altogether too high. Therefore it was found necessary to put another steam shovel on the surface and strip the rock mass. Rock quarried from all parts of the face was then relatively free from clay, the necessary amount of clay or shale being added at the cement plant.

Where it is desired to use the rock without admixture with stripping, the surface should be stripped back far enough from the face to prevent masses slipping into the pit. An instance was observed where soil had slipped into the quarry and was later removed by the slow and tedious hand-shovel and dump-cart method. It is much better to employ the most efficient stripping means possible, and to strip a fairly wide surface ahead of rock-quarrying operations.

In several quarries, notably in Illinois and Kentucky, a shale bed underlies good limestone, and when the limestone is removed in quarrying, the shale is rendered available.

CONDITIONS GOVERNING METHOD OF REMOVAL.

In choosing the cheapest method for stripping a number of factors must be considered. If a great quantity of material must be removed, rapid and efficient means, such as a steam shovel, should be employed, whereas a thin overburden may justify the use of more inexpensive equipment, although the cost per ton of stripping might be larger.

Much depends on the nature of the rock surface. If it has channels and pockets filled with clay or other material, removal of this material with machinery may be difficult, and the use of slower and less efficient hand methods may be necessary. Conditions become more complicated if the pockets contain masses of rocks.

The firmness or solidity of the overburden also influences the method. In some quarries the material may be readily removed with shovel, scraper, or clam-shell bucket without previous loosening, whereas in other places blasting may be necessary. The availability to a suitable dumping place governs to a considerable extent both the method of excavation and of transportation. Water supply and drainage are factors of extreme importance where hydraulic methods are contemplated.

METHODS EMPLOYED.

HAND LOADING INTO DUMP CARTS.

Hand shovels and dump carts are employed at some quarries where the amount of material to be removed is limited, and also at some quarries where the overburden is thick, but where the irregular rock surface discourages the use of steam shovels or other mechanical devices. In many instances clay could probably be removed more cheaply from large pockets with a clam-shell bucket than by hand loading. Hand loading is expensive, a cost of 30 cents or more a yard being common, hence this method should be avoided wherever possible.

TEAMS AND SCRAPERS.

Where a thin overburden must be removed from a wide surface and the dumping place is convenient, teams and scrapers may be used. Where removal to a considerable distance is necessary, the material is sometimes rehandled. At one Iowa quarry the soil was taken in scrapers to an elevated platform, dumped through an opening into dump carts, and hauled away with horses. By using wheel scrapers rehandling could have been avoided.

DRAG-LINE SCRAPERS.

A drag-line scraper equipment used for stripping in a Kentucky sandstone quarry is described in Bulletin 124.[a] An interesting modification of the method was observed in a Pennsylvania quarry. One of the conditions usually considered necessary for successful operation of a drag-line scraper is a near-by place of disposal, preferably at a lower level than the area being stripped. At this quarry the stripped material had to be hauled in cars to the dumping ground, which was some distance away. A single scraper of $\frac{7}{8}$-yard capacity, working on a drag line, stripped the soil back from the face. The loaded scraper was dragged up a plank incline onto a trap-door in the platform at the upper end of the incline. The trap door dropped under the weight of the load and dumped the soil into a car on a track below. The incline and the platform could be shifted laterally as required. Two men were employed to guide the scraper during loading, and part of the time of another man was required to control the cable and place the scraper in position. The entire operation of loading required only four men, and part of the time of a fifth. Except for the engine runner, the men were all unskilled laborers. The material was handled rapidly and the cost of loading was said to be about 3 cents a cubic yard. The loaded cars were hauled by cable to the dumping ground. The method seemed to be cheap and efficient.

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[a] Bowles, Oliver, Sandstone quarrying in the United States: Bull. 124, Bureau of Mines, 1917, pp. 29, 30.

In some quarries a scraper or bucket is lowered from the arm of a derrick and is loaded by hauling it toward the derrick with a horizontal cable wound on a drum. When full it is hoisted, swung over a car by the derrick arm, and dumped. No figures have been obtained indicating the efficiency of the method.

STEAM SHOVEL.

For removing large masses of material, a steam shovel is probably the most practical and efficient means. As the first cost of a steam shovel is high, to purchase one is unwise except where stripping is to be on a reasonably extensive scale. The nature of the rock surface also has a marked influence on the success of steam-shovel operations. In many limestone deposits the surface is eroded into deep channels and cavities which have been filled with soil. It is difficult, or impossible, to remove the material from such pockets with a steam shovel, also great difficulty may be encountered in laying tracks on the uneven surface. If the surface is irregular and the overburden is thin, a steam shovel should not be employed. On the other hand, if the rock has a smooth and uniform surface, a steam shovel may be operated efficiently, even if the overburden is only 2 or 3 feet thick.

The usual method of stripping with a steam shovel is to load the material into side-dump cars and haul them with a locomotive in trains of 6 to 20 cars to a near-by dumping ground. Loading into dump wagons has been noted, but a large number are needed to keep the shovel working. The use of wagons may be justified for a limited period of time if it is desired that the stripped material should be placed where laying a track is not feasible.

Waiting for empty cars is one of the chief causes of poor steam-shovel efficiency. This may be due to lack of cars or locomotives, or to inadequate trackage facilities. At one quarry visited, in order to avoid unnecessary loss of time, a train of empties stood on a siding ready to be shifted into place as soon as the loaded cars were removed.

Periods when cars are not available may be usefully employed in moving the shovel. In one New York quarry sufficient material for one train load could be reached from one position of the shovel. While the loaded cars were being hauled away, the shovel was moved to a new position and was ready for loading another train as soon as the empties returned.

Where the stripping is thin, much time may be lost in frequent moving of the shovel. When a full train load can not be obtained from one position of the shovel, poor efficiency must result if locomotives and train crews are thus kept in idleness. Under such conditions it may be wise to let the train crews work elsewhere while the steam shovel "casts over" a cut the entire length of the face. "Casting over" is a quarryman's term for making a cut with a steam shovel and depositing the material on the bank, thus forming a long ridge. When the next cut is made, a double amount of material is available from each position, and train crews are not kept idle while the shovel is being moved. Where the overburden is very thin, "casting over" may be repeated two or three times.

Usually the material overlying a rock deposit is soft enough to be handled readily with the steam shovel without blasting. In some localities, however, blasting is necessary. If large bowlders that require blasting are encountered, the rate of progress may be greatly reduced.

The cost of stripping with a steam shovel depends on the depth and nature of the materials to be removed, capacity of the shovel, condition of the rock surface, and facilities for removal of cars. Four cents a ton, actual working cost for loading only, is the minimum figure obtained. At one quarry having 1 to 10 feet of overburden, the total operating cost of stripping, including transportation, was 11 to 13 cents a ton. At an Illinois quarry having 20 to 30 feet of soil overburden, the operating cost of stripping, including transportation to a near-by dump, averaged 16 cents per cubic yard. Where the overburden is thin, the cost per ton will be relatively high, but the cost of stripping, per ton of rock obtained, will be relatively low.

A stripping conveyor operated in connection with a steam shovel by one Ohio company is noteworthy. The soil to be removed is 30 to 45 feet thick, and the bed of rock beneath is only 8 feet thick.

With the low efficiency of stripping observed in some localities, profitable quarrying under such conditions would be almost impossible. As the rock is removed, the area worked out is available for disposal of overburden. The equipment, as shown in Plate I, consists of an inclined bridgelike structure of steel mounted on wheels. On this incline, two independent cable cars on separate tracks are operated with an electric hoist. The cars are loaded with the steam shovel, hauled up the incline, and dumped. The total length of the conveyor is 175 feet, and it places the material 150 feet from the steam shovel. The shovel has a 2½-yard dipper, and each car is of 5-yard capacity. The whole machine may be moved forward as the shovel is advanced. The company claims that stripping can be done by this method at an actual working cost of 2 cents a cubic yard, not including overhead charges, depreciation, or interest on the investment, and that with a gang of 9 men 2,000 yards can be handled per day. The equipment is well adapted for stripping back into pits from which all serviceable stone has been removed.

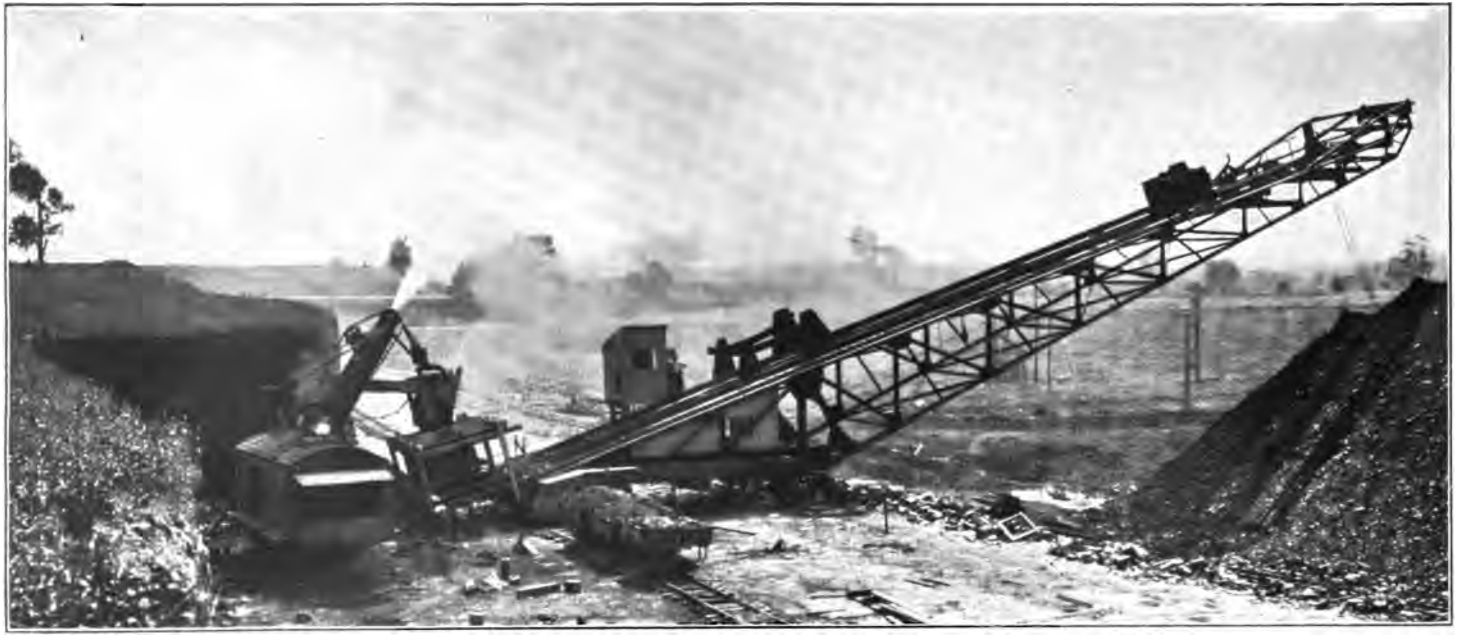
HYDRAULIC STRIPPING.

Hydraulic stripping is practicable only where there is an abundant water supply and good drainage. If the conditions are favorable it is one of the cheapest methods that can be used.

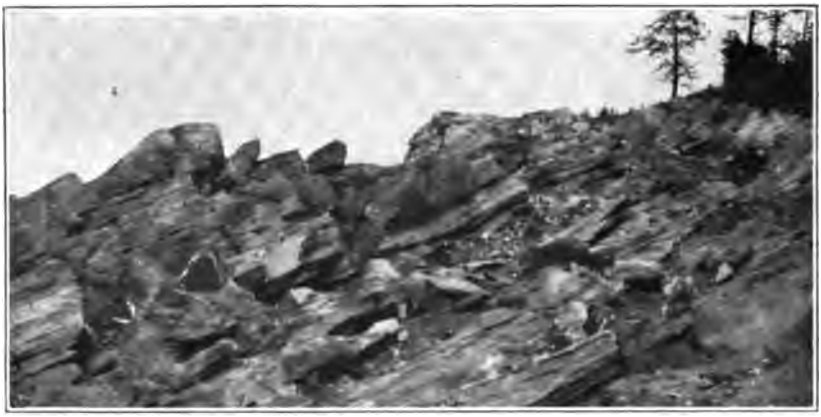
The usual method of hydraulic stripping is to pump the water through pipes and force it at high pressure through a nozzle directed against the bank. If the water has to be pumped some distance, it has been found advisable, in order to reduce the loss of power by friction, to use pipe of 4 or 5 inch diameter, close to the pumping station, and to reduce the size at successive intervals. The jet from the nozzle is usually so directed as to undermine the material and let it fall in a loose mass. It is then easily carried away in the stream. It is obvious that bowlders too large to be washed away retard the process, and if the material contains many bowlders the use of a steam shovel may be preferable.

In a quarry in northeastern Missouri where this method is employed a ravine affords a means of drainage back from the quarry face. As the material is rather hard holes 8 to 10 feet deep are bored with a soil auger, and the ground is loosened with small charges of explosive. The cost of stripping is said to be about 7 cents a cubic yard.

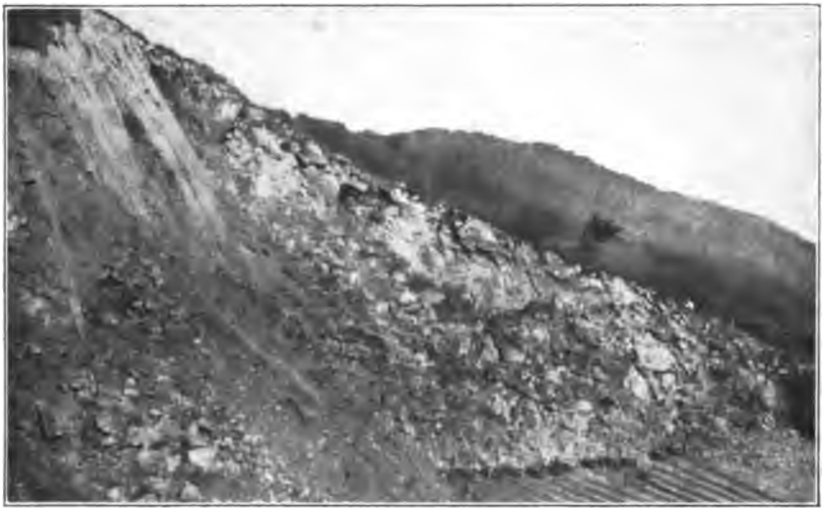
In an Indiana quarry stripping with steam shovels is supplemented with hydraulic stripping during seasons when the water supply is adequate. The material is washed down a channel, which is nearly parallel with the quarry face, and into a creek. A settling basin has been provided, but complaints are sometimes made of the turbidity



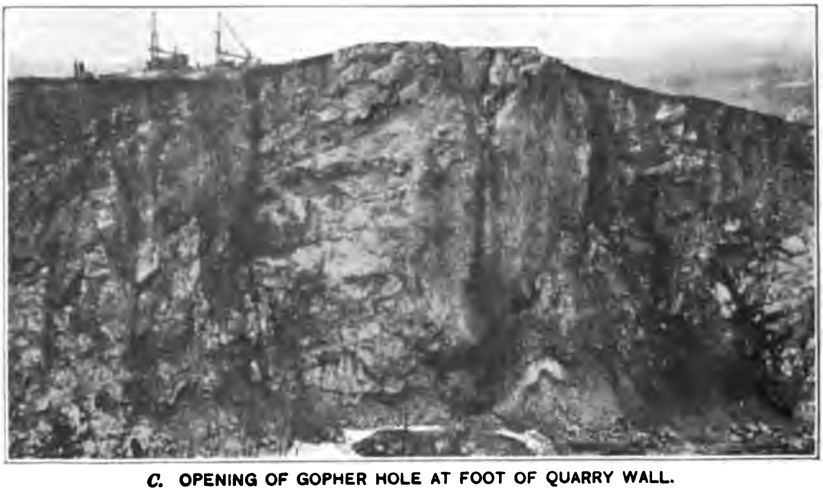
CONVEYOR AND STEAM SHOVEL USED FOR STRIPPING IN AN OHIO QUARRY.



A. ROUGH ROCK SURFACE EFFECTIVELY STRIPPED BY HYDRAULIC METHOD.



B. QUARRY FACE PARALLEL WITH STEEPLY INCLINED OPEN BEDS.



C. OPENING OF GOPHER HOLE AT FOOT OF QUARRY WALL.

of the stream below the basin. The effects of hydraulic stripping may be far reaching, and the operator must carefully consider a possible conflict with other interests.

In one quarry in eastern Missouri the conditions are unusual. The quarry is situated on a bluff facing the river. The water supply is therefore abundant, but the drainage is toward the quarry face, whereas in most localities where hydraulic stripping is practiced the water may be drained away from the quarry pit. Owing to this condition the soil-laden water must be conveyed through an aqueduct from the quarry face to the river bank. However, the operators claim that the cost of hydraulic stripping, including the construction and maintenance of the aqueduct, is considerably lower than with steam shovels.

At one Virginia plant, where the hydraulic method is employed successfully, the quarry faces a river. The overburden consists of red clay and sand, varying in thickness from 2 to 15 feet. Clay seams and clay pockets are abundant. Stripping is conducted during the winter months, when the resulting turbidity of the stream causes the least inconvenience to other industries. Work on rock quarrying is suspended during the stripping. The rock surface dips toward the river, and the overlying material is washed down onto the quarry floor. A row of barrels filled with earth placed near the river bank serves as a dam or weir to hold back the bulk of the stripped material, which would otherwise obstruct the stream. This material, after it becomes comparatively dry, is loaded into cars with a steam shovel and removed to a dump.

To those unfamiliar with the conditions in this quarry, it would seem more reasonable to remove the overburden with a steam shovel in the first place. The rock surface is, however, so deeply eroded that steam-shovel work would be impossible. Plate II, *A*, illustrates the extreme irregularity of the stripped surface. This view is an excellent example of the effectiveness of the hydraulic method in removing soil from erosion cavities.

ROCK WASHING.

In some localities quarrying is made difficult by open seams or pockets that extend many feet down into the rock. The material with which these erosion cavities are filled may be undesirable as a cement constituent. The hydraulic method of cleaning out such cavities is very successful where it can be employed. The usual method, however, is to clean the seams as well as possible by slow and laborious hand methods. The material that can not be thus removed is shot down with the rock. The subsequent sorting which is necessary makes it undesirable to use a steam shovel, and hand-loading methods are employed. When the good rock is removed the fine débris is loaded by hand into cars and removed to a dumping ground. The high cost of loading both rock and waste by hand would seem to justify the introduction of a more economical method.

The writer is of the opinion that in many quarries where such conditions are found a much better method would be to load with a steam shovel, taking rock and soil indiscriminately, and after crushing the material, pass it through washing screens. A sufficient water supply, of course, would be essential. The cost of installing a washing equipment would not be high. With sandy or other friable material the success of such a method would seem to be assured. It would not, however, work as successfully with sticky clay, which would be difficult to wash out. Also the crushing of rock mixed with clay may involve great loss of time in wet weather.

UTILIZATION OF STRIPPING.

As a rule the overburden of limestone deposits is of clay and can be used as one of the necessary constituents of Portland cement. Therefore in many localities part or all of the overburden is utilized for this purpose.

The material removed may also be used to fill in swamps, ravines, or other low places and thus render such areas available for agriculture or building. In one instance the reclaiming of swamp land for city lots brought in a return to the quarry company more than sufficient to pay the cost of stripping. The quarry operator may also utilize stripped material for making dams, roadways, or railway embankments on his property.

DRILLING.

RELATION OF DRILLING TO BLASTING.

Only certain phases of the subject of drilling can be considered independently of blasting, for the depth, size, spacing, and arrangement of drill holes are all related more or less closely to the character of the explosive and the method of blasting employed. In this chapter, therefore, the types of drills used and facts relating to their actual operation will be considered, other matters being reserved for subsequent consideration under "Blasting" (pp. 46 to 97).

PRIMARY AND SECONDARY DRILLING.

Two distinct processes of drilling are employed in practically all quarries where rock is obtained for cement manufacture. One, two, or more rows of holes are first drilled in the undisturbed bedrock, in order that heavy charges of explosive may be fired at various points within the rock mass to break it up for removal. The drilling of these holes is termed "primary drilling."

In most instances the primary shots do not break all the rock loosened into sizes suitable for handling, even with large steam shovels. The larger masses must, therefore, be broken up with subsequent shots. Whatever drilling is deemed necessary for these subsequent shots is termed "secondary drilling." The holes thus drilled are sometimes termed "pop holes."

PRIMARY DRILLING.

TYPES OF DRILLS EMPLOYED.

Primary drilling may be done with churn, wagon, tripod, electric, or hammer drills. The choice of drill depends on the nature and thickness of the rock ledge and on the method of blasting employed.

Churn drills.—The type of drill most commonly used in quarrying rock for cement manufacture is the ordinary well drill, also known as the cable or churn drill. Various types of churn drills are now on the market, the details of which may be easily obtained from trade catalogues. The diameter of drill hole used in cement practice varies from 4 to 6 inches. The rate of drilling varies according to the size of the hole and the hardness of the rock. In limestone, rates of 30 to 75 feet a day for 6-inch holes and 50 to 115 feet a day for 5-inch holes have been noted. In cement rock the rate is somewhat higher. The cost of churn drilling varies from 20 to 50 cents a foot. A fair average in moderately hard limestone is 30 cents a foot for 5⅝-inch holes. A cement company near Glens Falls, N. Y., claims that the cost of drilling 6-inch holes with an electric-driven churn drill, which required only one man to run it, was less than 15 cents a foot, including repairs.

Wagon drills.—The so-called wagon drill is of the reciprocating type, operated by steam or compressed air. It makes a hole 3 to 4 inches in diameter. The machine may be so rotated as to drill at either side or in front of the wagon upon which it is mounted. The common method is to drill a hole at each side of the wagon from each position, and thus carry two rows of holes simultaneously. The drill has a long vertical range; hence the steel does not need to be changed so frequently as in tripod drilling.

Tripod drills.—Tripod drills, with which every quarryman is familiar, are used to some extent in cement practice, although during recent years they have largely given place to churn drills. They are used most commonly in unsound rock, in thin ledges, or for drilling horizontal or so-called "snake" holes (see p. 57) at the base of the quarry ledge. Common sizes of tripod drill holes are 1¾ to 2¾ inches. The "electric air drill" is used successfully in some localities. Air is supplied to it through flexible hose from a portable motor-driven compressor, which may easily be moved whenever the drill is moved to a new position. An average cost of tripod drilling in limestone is 15 cents a foot.

Electric drills.—Motor-driven electric drills give satisfactory service, especially in rock mining.

Hammer drills.—The larger hammer drills in common use are nonreciprocating drills, operated without a tripod, and are provided with an automatic rotating device. They use hollow steel, through which the exhaust air passes and blows the cuttings from the drill hole. They are employed in primary drilling in some quarries with low benches not exceeding 8 or 10 feet in height. They are also used for making shallow holes to supplement churn-drill holes where an unusually hard bed of rock occurs at a short distance below the surface. In the opinion of a number of quarrymen they drill more rapidly than tripod drills, and as one man only is required for each drill they are less expensive to operate than the machines requiring both a driller and a helper. A drilling rate of 180 feet per day in limestone has been noted. The hammer drill works most successfully when run dry, as the presence of a small amount of water in the hole may make the cuttings muddy so they can not be blown out with the exhaust.

EFFECT OF ROCK STRUCTURES ON DRILLING.

The presence of numerous open joints in a rock ledge, particularly if the joints are irregular, complicates drilling. The drill may be deflected by running into a fissure, thus making drilling slow and difficult. However, the chief objection is that a drill hole penetrating an open seam or cavity is usually useless, as the gases generated by the blast will escape into the open spaces and the shot will be relatively ineffective. Usually such a hole is abandoned and a new one made. Many of the holes drilled in unsound rock are for this reason never used.

ADVANTAGE OF PLOTTING JOINT SYSTEMS.

Where the joints are straight and regular and can be observed on the rock surface, the driller can usually avoid them by using good judgment. If the upper strata are extremely shattered, it may be impossible to detect the joints on the surface, and the driller must judge their position by observing the vertical face. If their trend is fairly constant, their position at a distance of 20 or 30 feet back from the face may be estimated with a reasonable degree of accuracy. But if the joints are irregular or curving, or tend to change abruptly in direction, the placing of holes may become mere guesswork, and many useless holes will be drilled. In one quarry with a 50-foot face, where tracing the probable curves of open joints was a difficult matter, the writer was informed that one driller had in three weeks drilled only two serviceable holes, all others having been abandoned when partly completed on account of their penetrating clay pockets.

A detailed description of the origin and nature of joints is given in Bulletin 106.[a] It is pointed out that in general joint planes tend to occur in parallel systems, and the planes of each system tend to be more or less constant in direction. By making a careful and systematic measurement of the joints with compass and clinometer and plotting them carefully on paper, it will usually be found that they have a systematic arrangement and are less irregular and haphazard than they appear to be from casual observation. A chart of all the joints on which definite observations can be made in a given deposit would, without doubt, be of great assistance to the driller in choosing a proper location for drill holes, and much useless drilling would thus be avoided.

KEEPING DRILL HOLES CLEAN.

On account of the modern method of firing a large number of drill holes at a single blast, several months may elapse from the time a hole is drilled until it is loaded. If proper precautions are not observed, the hole may become partly filled with débris, which must be cleaned out. A good plan, therefore, is to drive a wooden plug into the hole as soon as it is completed. Where the upper part of the ledge is thin-bedded, loose rock, it may not be possible to plug the holes effectively without casing them for a depth of several feet. Sheet-metal casings may be made at low cost and are highly recommended for use in deposits of this type.

CLEANING DRILL HOLES.

The method used by one Oklahoma company for removing débris from holes before loading is to place a gas-pipe tripod over the hole, and by means of a rope and pulley lower a section of gas pipe into the hole. By repeatedly raising the pipe several feet and allowing it to drop, it gradually sinks into the débris. Compressed air is then forced through the pipe and blows the refuse out. If the material is too coarse to be blown out, it is first cut up with a steel bar. As the cost of cleaning some drill holes has exceeded the original cost of drilling, all holes recently drilled by this company are cased at the top and carefully plugged.

SECONDARY DRILLING.

For secondary drilling, that is, the drilling of blocks that were insufficiently broken by the primary shot, hammer drills are most commonly used. The lines for supplying compressed air to the primary drills are usually laid on the quarry bank adjacent to the face. Air for secondary drilling is usually obtained from these lines by means of flexible hose. In one Iowa quarry the air is supplied by a small compressor mounted on the steam shovel. In a New York quarry a motor-driven compressor is mounted in a box car and hence can be readily moved to near the point where the hole is to be drilled.

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[a] Bowles, Oliver, The technology of marble quarrying: Bull. 106, Bureau of Mines, 1916, pp. 22–29.

BLASTING.

IMPORTANCE OF BLASTING.

In the blasting of rock for use in making cement, breaking into fragments that can be easily loaded into cars and can be handled by the crushers at the cement plant is essential. Because of the importance of blasting and the cost of explosive, no phase of the quarry industry under consideration is receiving more attention by investigators at present. As blasting represents such an important item in the cost of quarrying, evidently the introduction of efficient blasting methods at any quarry may materially reduce the total quarry costs. The wide variations in the character of rock deposits in different localities make it impossible to formulate rules for successful blasting that will uniformly apply. However, an attempt will be made to point out the methods followed in various localities and the conditions under which the various methods may give the best results.

PRIMARY AND SECONDARY BLASTING.

Blasting, like rock drilling, may be divided into primary and secondary processes. The term "primary blasting" refers to the shots by means of which the quarry ledge is broken into fragments of various sizes. "Secondary blasting" includes whatever shots may be necessary to break up the larger masses resulting from the primary shots into fragments small enough for loading.

PRIMARY BLASTING.

COMPLEXITY OF PROBLEM.

Primary blasting is one of the most complex problems with which the quarryman has to deal, owing to the many variable factors and the multitude of ways in which these factors may be combined. The chief variable factors are: (1) Height of face; (2) hardness and uniformity of rock; (3) attitude of beds; (4) prevalence of open bedding seams and joints; (5) attitude of quarry floor; (6) size and depth of drill holes; (7) arrangement and spacing of drill holes;

(8) number of holes shot at one time; (9) size of charge; (10) position of charges in drill holes; (11) type of explosive used; (12) method of firing shots; (13) method of loading rock; (14) size of crusher.

MODERN TENDENCIES IN BLASTING.

During recent years the tendency among quarrymen has been to throw down a large mass of rock at a single shot. Formerly the method of quarrying a ledge in successive benches or steps was widely employed. Of recent years, however, this method, which may be termed "multiple bench quarrying," has been generally abandoned, and most quarries are worked as single benches, even to depths upward of 150 feet. As a result, churn drills have been substituted for tripod drills in most quarries. This tendency has been increased by the wide use of steam shovels for loading. With a steam shovel rock may be loaded rapidly, and much larger fragments can be handled than was possible under hand-loading methods.

ADVANTAGES OF SINGLE BENCH.

One of the chief reasons for single rather than multiple bench quarrying is the increased efficiency attained by throwing down larger masses of rock at a single shot. Also, when several benches are worked it is necessary to clean the upper benches after shooting, and this cleaning requires a great deal of time. In quarrying with a single bench no bench cleaning is necessary. Another advantage is the increased safety which results. If quarrying is conducted on several narrow benches rock may fall from the upper benches and injure workmen on the lower levels. However, where benches are wide this danger may not exist.

DANGERS OF SINGLE BENCH.

In some quarries single-bench quarrying with churn drills has resulted in the development of vertical faces 80 to 150 feet high. The danger from loose rock falling many feet is obvious. On account of this danger, in some European countries the height of the face is restricted by law. Where high faces are maintained great care should be exercised in cleaning down all loose material from the face and from the rock surface close to the edge of the pit. In quarries where the rock is unsound men may be suspended with ropes at the face after each shot, to bar or blast down all loose material.

THE USE OF CHURN DRILLS.

In operating quarries as single benches many feet in height churn drills are widely used. In judging the efficiency of drills one must keep in mind the prime object of drilling. Drill holes are made for the purpose of placing in the proper position enough explosive to break the rock efficiently. It is regarded as a necessary condition for successful blasting that a considerable part of the charge shall be near the bottom of the mass to be moved. For making deep holes churn drills are found to be more efficient than tripod drills. In tripod drilling the steel has to be changed frequently and long drills are difficult to handle. Moreover, tripod drill holes are rarely more than 2¾ inches in diameter at the top, and the decrease in gage as successive drills are used makes the deep holes much smaller in diameter at the bottom than at the top. As more explosive is required at the bottom of the hole than near the top, for practical purposes the hole is upside down. Hence, to shatter the rock efficiently in tripod drilling the holes must be drilled close together or else the holes must be "sprung" or "chambered"; that is, enlarged at the lower end by discharging small amounts of explosive in them. In springing, a small charge is used at first, and for each successive charge a larger quantity of explosive is employed. Commonly, about one-half pound of "40 per cent" dynamite is used for the first charge, the amount being doubled for each succeeding charge. The process is slow, requires a good deal of explosive, and is dangerous.

Churn drills, on the other hand, make holes 4 to 6 inches in diameter, and there is no loss of gage. Thus, the cost per foot of hole drilled may be much higher than that of tripod drilling, but the holes may be spaced much farther apart and still admit sufficient explosive to do the necessary work without springing.

In this connection the driller must keep in mind the proper relationship between volume and diameter of drill holes. Many drill men are under the impression that as a 6-inch hole has twice the diameter of a 3-inch hole, the former will hold twice as much explosive per foot as the latter. The volume, however, varies as the square of the diameter, hence a 6-inch hole holds four times as much as a 3-inch hole, a 5-inch hole holds 2$\frac{7}{9}$ times as much as a 3-inch hole, etc. When the driller realizes, as most drillers do and as all drillers should, that doubling the diameter of the drill hole increases its capacity for explosive four times, he will better appreciate the advantage of a churn drill for deep-hole blasting.

Springing of churn-drill holes to increase their capacity for explosive has been employed in some mining operations where a heavy toe is to be removed, but is not commonly employed in ordinary quarry practice.

The efficiency of large holes may be shown in another way. Let us assume that it has been found by experience that blasting in tripod-drill holes is effective if the holes are 10 feet apart and have 10 feet of burden, that is, are 10 feet back from the face; whereas churn-drill holes would give equally good results when spaced 20 feet apart and with 20 feet of burden. In order to shatter a mass 200 feet long and 20 feet wide, 11 churn-drill holes would be sufficient, whereas 42 tripod-drill holes would be necessary, as shown in figure 2. If the face is 30 feet high, the total mass of rock moved would be 200×20×30, or 120,000 cubic feet; that is, about 10,900 cubic feet for each churn-drill hole, and only 2,850 cubic feet for each tripod-drill hole.

In general, the churn drill is not satisfactory for quarrying thin ledges of rock. Thin ledges require small charges. When a small charge is placed in a large hole, too great a proportion of the explosive is concentrated at the bottom of the hole. Moreover, it is obviously a waste of time and power to drill a larger space than is necessary to hold the explosive. In general the use of churn drills for benches less than 25 or 30 feet high is not considered wise.

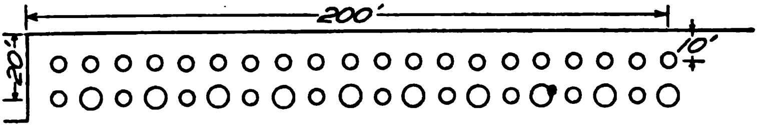

FIGURE 2.—Diagram showing relative efficiency of churn-drill holes and tripod-drill holes. Large circles represent churn-drill holes; small circles, tripod-drill holes.

THE USE OF WAGON DRILLS.

The wagon drill is capable of drilling to a maximum depth of about 40 feet. It drills smaller holes than are usually made with the churn drill, but drills much more rapidly. Experience has shown that in some quarries the rock breaks better for a given quantity of explosive if the holes are smaller and spaced more closely than churn-drill holes are usually placed. Therefore, wagon drills are well adapted for drilling in such rock.

The wagon drill may also be used with success on thin ledges where, as pointed out in a previous paragraph, churn drills can not be used to advantage.

THE USE OF TRIPOD DRILLS.

Tripod drills are commonly employed in quarries where the rock is worked in one or more low benches. Where such drills are used, benches are rarely more than 16 feet high. The multiple-bench method, which has recently given place largely to single-bench quarrying, is still practiced in some localities because the advantages of using the churn-drill are not realized. In other places, lack of capital prevents the purchase of the new equipment necessary to make the change, although the advantages of the more modern methods are admitted.

In some quarries, however, it is claimed that the conditions justify the use of tripod drills. These conditions are discussed in the following paragraphs.

ADVANTAGE OF USING TRIPOD DRILLS FOR UNSOUND ROCK.

As pointed out on page 44, the presence of irregular seams or pockets filled with clay greatly complicates drilling, because these are often difficult to avoid in placing the holes. Mere avoidance of seams is, however, only part of the problem. The holes may all be in sound rock and yet be so arranged that the blast will be ineffective. In general, it may be stated that in seamy rock each individual mass bounded by open seams should have at least one hole drilled in it, and, if the mass is small, a small hole is sufficient. The reason for this statement is given in the following paragraph.

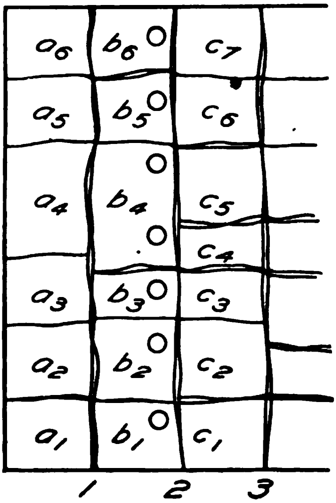

FIGURE 3.—Arrangement of drill holes in seamy rock.

Suppose that a series of open seams parallel the quarry face and are spaced 10, 10, and 30 feet, respectively, from the face, as shown in figure 3. An average burden for churn-drill holes is 18 to 25 feet. If, in this instance, churn-drill holes are placed 18 feet from the face, what will be the effect of the shot on the masses designated a_1 to a_6, between the face and seam *1*? It is a well-known fact that an open seam acts as a cushion and almost or entirely checks the shattering effect of an explosion. The masses b_1 to b_6, between seams *1* and *2* would without doubt be shattered normally, as the drill holes are situated in these masses. However, the masses a_1 to a_6, protected by the open seam *1*, would not be shattered much and would merely be pushed out. If the drill holes were placed 25 feet from the face, they would be in the masses c_1 to c_7, and obviously the masses designated a_1 to a_6 and b_1 to b_6 would receive little of the shattering effect. A blast of this type, which pushes out great blocks weighing many tons each, would be very inefficient, as an excessive amount of secondary blasting would be necessary.

Therefore, in order to effectively shatter rock intersected by open joints, such as are represented in figure 3, at least one drill hole in each individual mass seems to be necessary. Moreover, the hole should be so placed that the shot is balanced, thus promoting equal effectiveness in all directions from the hole. Holes drilled in the masses designated a_1 to a_6 should be nearer to seam *1* than to the face, for the rock near the open face offers less resistance to the explosive force than that near the earth-filled seams at the back. The proper position of the hole can not be stated arbitrarily; it depends on the character of the rock, and can be determined only by experiment. However, on the assumption that the proper position is 7 feet from the face and 3 feet from the seam, it is obvious that a churn-drill hole 7 feet from the face would be too large. The small amount of explosive necessary to break a mass 10 by 12 feet or 10 by 15 feet in area could be placed in a hole much smaller than that ordinarily made with a churn drill. Consequently, drilling a large hole in such a position would be a waste of time and labor.

For seamy rock one of the smaller types of drills is to be preferred, and the tripod is commonly employed. Thus multiple-bench quarrying may be the best method to use in unsound rock. When churn drills are used in rock having steeply inclined open seams, the drill bit tends to follow the seam and thus make further drilling difficult or impossible. This involves long delays, and may necessitate the abandonment of partly completed holes. Instances have been noted where many feet of drilling are wasted in this manner every month. Some quarrymen claim that this difficulty is less pronounced with tripod drills. For those skilled in the use of churn drills, however, the difficulty is no greater than where tripods are used. A small charge of explosive is sometimes used in straightening a drill hole that crosses an inclined open seam. Some operators drop pieces of scrap iron in the hole to delay drilling until the undesirable shoulder is cut away.

USE OF TRIPOD DRILLS IN OBTAINING A UNIFORM MIX.

It has been stated that the chemical composition of rock quarried for cement manufacture must be carefully taken into account. The rock is continually being sampled and analyzed as it enters the plant and proportioned in the way that will give the best product. If the rock in the quarry ledge is uniform in composition, it is a simple matter to obtain a mixture of the desired composition, and the method of quarrying has little or no effect on the mixing. However, if the rock varies in successive beds the method of quarrying has a marked influence on the task of maintaining a uniform mixture. This is particularly true of certain cement-rock quarries of the Lehigh district, where, by advantageous handling of rock, a proper mixture can be obtained from one ledge without the addition of either limestone or shale from any other locality.

Let us assume that the beds are flat-lying and that they vary considerably in their calcium content. If the upper beds have a low and the lower beds a high calcium content, quarrying from the upper beds alone would shortly give a lean mixture, and quarrying of the lower beds alone would give one too high in calcium. A method employed by one company having such a quarry is to work it in two benches, the upper bench including the low-calcium stone and the lower bench the stone higher in calcium. As these benches are low, tripod drills are employed and the rock is loaded into cars by hand. As the rock is hauled to the crusher the number of carloads from each of the benches is proportioned according to their relative composition so as to give a proper mixture.

A method of shooting down the entire quarry face, using churn-drill holes, has been suggested. Some quarrymen state that by properly balancing the charges in the holes the material from the upper and that from the lower beds may be mixed fairly well as the rock is shot down, and that the rock then may be handled much more efficiently with steam shovels than by hand methods.

The strata may not, however, be flat-lying, but may be steeply inclined, as in many limestone and some cement-rock quarries. In several quarries having steeply inclined beds the rock is drilled with tripod drills, in benches 8 to 16 feet high, and after being shot down is loaded by hand. The quarry face is perpendicular to the strike and intersects many beds. Rock is loaded from various points along the face at the same time, and a constant mixture is thus maintained. A mixture is sought which, with the addition of sufficient shale to give the proper ratio of calcium to aluminum, will have the proper proportion of silica and a minimum of magnesium.

The objection raised to the use of a steam shovel in such quarries is that it loads for a considerable time from one position and thus prevents proper mixing of the various types of rock. For example, the shovel might work for a period in rock having only 65 per cent of $CaCO_3$, which would be too low in calcium for a cement mixture, and during another period in rock having 85 per cent of $CaCO_3$, which would be too high. For this reason, a number of quarrymen recommend hand loading for variable rock.

It is a generally recognized fact that blasting rock in low benches and loading it by hand is a costly method of quarrying, hence the devising of means for overcoming any difficulties that may stand in the way of churn-drill and steam-shovel methods is desirable. Consequently the following suggestions are made regarding improved methods of quarrying steeply inclined beds that vary in composition.

In the methods described in preceding paragraphs quarrying is so conducted that the rock may be selected proportionally from each of the variable beds. In steeply inclined beds, selection is made possible by cutting across the beds horizontally. If a high face is developed instead of long, low benches, approximately the same mixture may be obtained vertically as is now obtained horizontally. This is illustrated in figure 4. The beds that differ in composition are designated *a*, *b*, and *c*. The quarry face is bounded by the lines *1* and *2*. If deep quarrying and steam-shovel loading were pursued the source of the material available from one position of the shovel may be represented as bounded by the vertical lines *3* and *4*. As may be seen from the figure, the same beds would be intersected as by the other method. From other positions to right and left of the one represented variations in thickness may occur, giving more material from certain beds and less from others, but in general a fairly representative mixture can be obtained.

For both flat-lying and inclined beds the objection may be raised that the use of churn drills and steam shovels will not permit sufficient latitude in varying the proportion of stone from each bed, for the steam shovel must take all material shot down, irrespective of its composition. If the various beds differ markedly in composition, this objection may be valid. It is suggested, however, that difficulties caused by local variations in composition may be largely overcome by using a large storage bin, where the rock may be distributed in such a manner as to equalize all variations. This matter is discussed at greater length in subsequent pages where the preparation of the raw mix is considered.

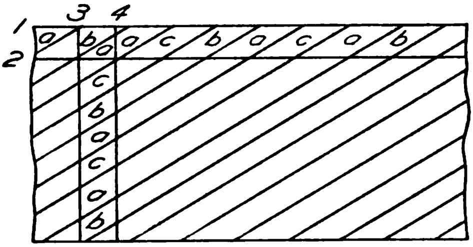

FIGURE 4.—Method of working steeply inclined beds to obtain uniform rock mixture; *a*, *b*, and *c* are beds of rock differing in composition.

USE OF TRIPOD DRILLS WHERE THE ROCK CONTAINS POCKETS OF CLAY.

In certain quarries, clay pockets are prevalent in the upper beds and separation of the clay from the rock may be desirable. If the entire face were shot down by means of blasts in churn-drill holes, the clay would be mixed with the rock, which would make the removal of both clay and rock difficult, especially in wet weather. Therefore, in some quarries it is regarded as a better plan to drill the upper bench with tripod drills, and shoot it down. When this material is removed the lower solid ledge may be shot down.

Usually, however, the clay may be used as a constituent of the cement, then, if it can be done successfully, quarrying rock and clay together is much better than to separate them and later mix them together again. One objection to handling them together is that in wet weather the clay makes the rock muddy and difficult to crush. With roll crushers, less difficulty is encountered than with gyratory crushers. Another objection is that the steam shovel may at one time be loading material that is largely clay and at another time almost pure rock, and thus complicate the problem of maintaining a uniform mix. This difficulty may be minimized by having a large storage bin.

Another method of obtaining a uniform mixture of clay and rock is well illustrated in an eastern Missouri quarry. In this quarry the upper beds contain many clay seams, and a face 50 feet high is shot down with blasts in churn-drill holes. The charge is so adjusted that the clay is thrown out evenly over the whole mass. When the material is loaded with a steam shovel, the rock and clay are found to be proportioned with a fair degree of uniformity.

THE USE OF HAMMER DRILLS.

Hammer drills may be substituted for tripod drills wherever the use of the latter seems to be justified, for drilling holes ranging up to 12 or 15 feet deep, and their successful use for even greater depths is claimed. They work most successfully where the rock is free of water. If resort to bench quarrying is deemed advisable, hammer drills should be substituted for tripods wherever possible. A drill of this type may be operated by one man, and 200 to 250 feet of hole per day has been attained. To obtain the highest efficiency in the use of hammer drills, the air pressure should be not less than 90 pounds. Hammer drills are used widely for secondary drilling.

ARRANGEMENT OF CHURN-DRILL HOLES FOR MULTIPLE SHOTS.

The arrangement and spacing of drill holes is an extremely variable factor. In most primary blasting, a series of holes are fired simultaneously by electricity. Such a series is termed a "battery of holes" or a "multiple shot" to distinguish it from a "single shot," where only one hole is discharged at a time. For multiple shots the holes are drilled in one or more rows. The distance of the row of holes from the face is termed the "burden," and the distance apart of the holes in the row is termed the "spacing." Where several rows of holes are shot at one time, the burden of the second and following rows is the distance between that row and the next succeeding row toward the face. In some instances the holes of the second and following rows are immediately behind those in the first row, as illustrated at *A*, figure 5. Usually, however, the holes are staggered, as illustrated at *B*, figure 5. Generally better results may be obtained by staggering the holes than by placing them in straight rows.

Difficulty is sometimes experienced in properly measuring the burden from an uneven face. Usually when the burden is heavy the effect of the blast is to shatter and blow out the surface rock for some distance back of the line of drill holes. Such "break-backs" occur most commonly where the upper beds are unsound or where the charge is placed too high in the drill hole. In placing the next line of holes the burden should not be measured from the edge of this slanting face, because it would obviously be too heavy for the lower part of the face, but should be measured from the position of the previous line of holes. In some quarries where holes are shot in single rows, when one row of holes is completed the burden of the second row is measured and the positions for the holes marked before the first row is fired. Thus the difficulty of measuring from an uneven face is eliminated.

The burden and spacing of drill holes varies according to the nature of the rock. As a rule, the tougher the rock the closer the spacing should be. If the toughness varies from point to point along a rock ledge the spacing of holes for a multiple shot should be varied accordingly.

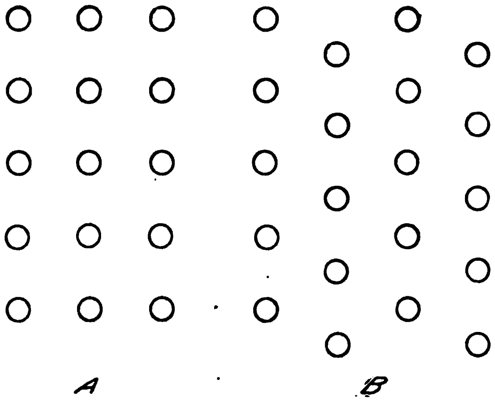

FIGURE 5.—Methods of placing drill holes for multiple shots: *A*, Holes in parallel; *B*, holes staggered.

In order to determine the proper spacing for drill holes in any particular quarry, one blasting engineer recommends beginning with a close spacing and then gradually increasing the distance until the maximum spacing that will properly shatter the rock is attained. In a 35-foot ledge of moderately hard limestone, drill holes should have a spacing of about 12 feet and a burden of 15 feet. The spacing and burden should increase gradually with increasing depth of holes. Where holes are 100 to 150 feet deep the spacing should rarely exceed 20 feet or the burden 25 feet.

The arrangement of drill holes for typical multiple shots in a number of quarries throughout central and eastern United States is given in Table 3.

TABLE 3.—*Kind of rock, number, depth, and arrangement of churn-drill holes for typical multiple shots.*

Kind of rock.	Average depth of holes.	Number of holes.	Number of rows.	Opposite or staggered.	Burden.	Spacing.
	Feet.				*Feet.*	*Feet.*
Cement rock	145	10 to 22	1		20	18
Do	80	3 to 6	1		20	20
Do	85	20	2	Staggered	18 and 18	18
Limestone	50	50 to 60	1		15	15
Do	30	20 to 30	2	Opposite	12 and 12	12
Do	22	52	2	Staggered	10 and 10	12
Do	40	26	1		18	18
Do	55	50	2	Staggered	15 and 15	15
Do	75	30 to 50	2	do	15 and 14	18
Do	19	41	2	do	8 and 7	12
Do	85	18 to 25	2	do	20 and 5	20
Do	40	10 to 15	1		20	12 to 15
Do	15 to 20	40 to 60	3	Staggered	8, 5 and 5	7 to 8

Of the examples in Table 3, wherever more than one row of holes is employed, the holes are staggered in every instance except one. Some have the holes arranged in three, four, five, or even six rows, in all these the adjacent rows are staggered. Where two rows of holes are used the burden of the second is in some examples equal to that of the first, whereas in others it is less than the first. It is regarded better practice to make the burden for the second and subsequent rows one-half to two-thirds that of the first row.

No attempt is made to judge the relative efficiency of the various arrangements recorded. In fact, to do so would be almost impossible, because the rock varies so widely in different localities that no one method is applicable to all types. Possibly the arrangements shown are not the most efficient that could be used. However, they represent practice at quarries that have been in operation for many years and where various methods have been tried, consequently it may be assumed that a fair degree of efficiency has been attained.

ARRANGEMENT OF SMALL DRILL HOLES FOR MULTIPLE SHOTS.

Wagon-drill holes may be fired in a single row or in several rows. They are spaced much more closely than churn-drill holes. Holes made with a tripod or hammer drill may be fired in one or in several rows, and the burden and spacing are usually not more than 6 feet.

ATTITUDE OF BEDS IN RELATION TO BLASTING.

ADVANTAGE OF FLAT-LYING BEDS.

In blasting rock a certain proportion of the explosive force is expended in separating the mass from the quarry floor, the proportion required depending on the ease of separation. Hence an open bedding plane at the floor greatly facilitates separation and promotes efficiency in blasting. Where the rock lies approximately flat and where open bedding planes occur at intervals, the quarryman should endeavor to make one of the open seams the quarry floor.

DIFFICULTIES ENCOUNTERED IN STEEPLY INCLINED BEDS.

If the beds are steeply inclined, the quarry floor must of necessity cut them, consequently open bedding seams can not be utilized to make a smooth quarry floor. In such quarries, breaking the rock free from the quarry floor requires considerable force. The burden of material between the bottom of the bore hole and the face is termed the toe. It is always desirable to obtain a "clean toe," by which is meant sufficient shattering of the material that constitutes the toe to make its entire removal possible without excessive secondary blasting. When there is no plane of separation at the quarry floor it is customary to drill the holes 2 to 5 feet below the floor level, in order that enough of the explosive may be concentrated at the base to give a clean toe.

If insufficient explosive is used or drill holes are improperly arranged, masses of solid rock may remain projecting into the quarry at the toe and cause great difficulty in obtaining a clean toe for the next shot. Under such conditions the churn-drill holes are as a rule supplemented with "snake holes." The term "snake holes" is applied to holes drilled in an approximately horizontal direction and usually near the quarry floor.

DIRECTION OF QUARRY FACE FOR STEEPLY INCLINED OPEN BEDS.

Where rock is quarried from steeply inclined beds with open bedding planes at intervals, the main quarry face may be parallel or be perpendicular to the strike. The plan that tends to develop the quarry face in the direction that results in the most efficient blasting should be followed.

When the quarry face is parallel with the strike and the beds dip toward the face, the rock tends to separate along the bedding planes and slide down, resulting in a steeply inclined rather than a vertical face, as illustrated in Plate II, *B* (p. 40). An unfavorable feature of blasting a sloping face is that the burden is heavier at the toe and the churn-drill holes have to be supplemented with "snake holes." Plate III, *A*, illustrates the drilling of snake holes in a steeply inclined face. This supplementary process adds to the cost of both drilling and loading holes. Under normal conditions of quarrying with a vertical face, snake holes are rarely required. Another objection to a slanting face is the danger to which quarrymen are exposed from the rock sliding down the smooth, steep slope. If the face is maintained more nearly vertical than the dip of the beds, the danger is increased.

Also, where the beds dip toward the face the rock tends to slide down in large masses not sufficiently shattered, especially when low-grade explosives are employed. Under such circumstances it is best to use a high-grade rapid-detonating explosive, which will act more quickly and effectually shatter the mass before it begins to slide.

If the quarry face parallels the strike and the beds dip away from the face, a still more unfavorable condition prevails. This condition is illustrated in figure 6. As the face is continually advanced in the direction of the dip, the rock must be forced up the dip, an operation that requires an excessive amount of explosive. Furthermore, "snake-hole" blasting, if employed, tends to undermine the face and render it unsafe. Also, there is usually great difficulty in obtaining a clean toe. This method of quarrying has been observed by the writer in three localities, and in each instance the costs were excessively high.

When the quarry face is perpendicular to the strike, as illustrated in Plate III, *B*, a vertical face may be maintained, and the rock is thrown down with comparative ease. If the rock has open bedding seams, part of the force of the explosive may be lost. However, this loss will occur to some extent whatever the direction of the face. If all the factors are taken into consideration it is evident that a quarry can be developed to best advantage with the face crossing the strike.

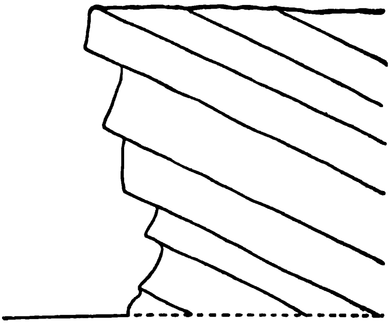
FIGURE 6.—Diagram showing conditions where rock beds dip away from quarry face.

BLASTING IN CHAMBERED "SNAKE HOLES."

Under certain conditions blasts in chambered "snake holes" are found to give the best results. This method is employed in the steeply tilted irregular limestones of eastern Pennsylvania. The method could probably be used to advantage in some ledges consisting of steeply inclined beds with open seams, as, for example, those near Security, Md., and Spear's Ferry, Va. If the face exceeds 60 feet in height the snake holes should be supplemented by vertical holes, or the rock should be removed in two benches with snake holes.

The snake holes are drilled with tripod drills and are 24 to 30 feet deep. It is most convenient to start the hole about 2 feet above the quarry floor and to slant it enough to reach a little below floor level at its inner end. As recommended by S. R. Russell,[a] blasting en-

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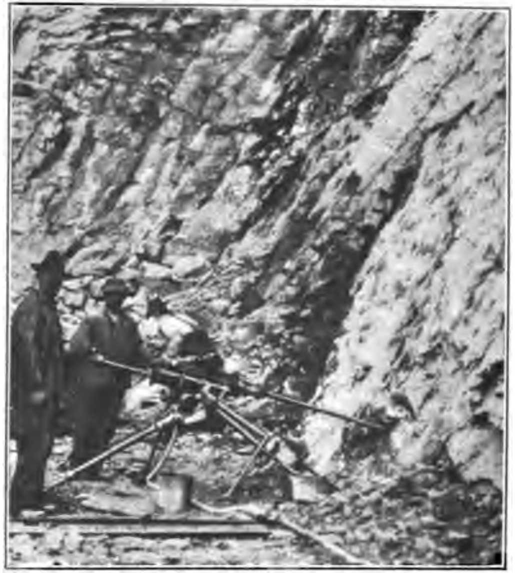
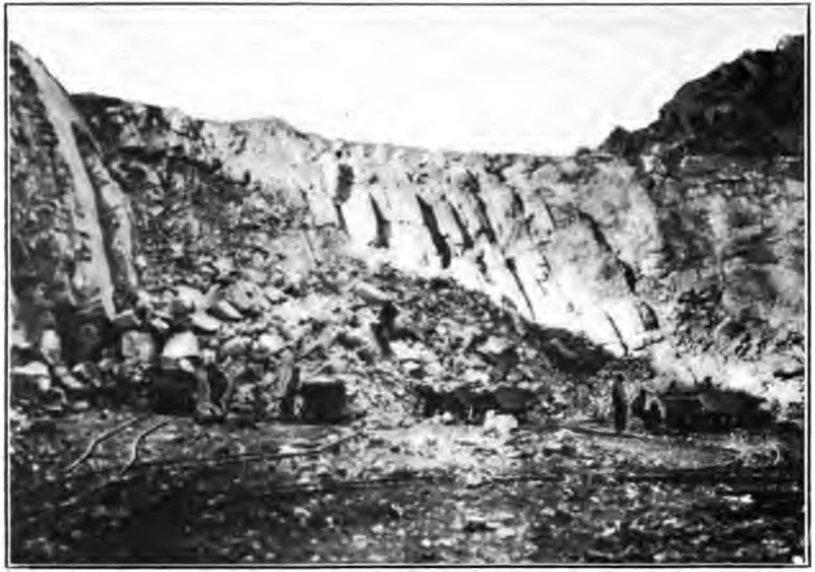
[a] Russell, S. R., Modern quarrying: Du Pont Magazine, June, 1916, p. 9.



A. DRILLING "SNAKE" HOLES IN A WEST VIRGINIA QUARRY.



B. QUARRY FACE MAINTAINED PERPENDICULAR TO STRIKE OF STEEPLY INCLINED OPEN BEDS.

gineer of E. I. Du Pont de Nemours & Co., the holes should be sprung about five times to give chambers of sufficient size. The first springing charge should consist of two 1½ by 8 inch cartridges of dynamite, the second six cartridges, the third 12 to 15 cartridges, the fourth 20 to 30 cartridges, and the fifth 40 to 50 cartridges. A mark is placed on the tamping stick, and after each springing shot the chamber is filled to the mark on the stick. As a safety precaution a period of at least two hours should elapse between springing shots in order to allow the rock to cool. Holes are usually so sprung that they can each be loaded with 100 to 300 pounds of explosive. The charge should not extend more than 4 or 5 feet from the chambered part into the shank of the hole. "Forty per cent" dynamite gives satisfactory results in limestone. A convenient device for loading snake holes is a tin or brass pipe, which can be pulled out gradually as the explosive rises in the chamber. This method of blasting is cheap and efficient where conditions are favorable, although considerable care and judgment are required of the blaster.

"GOPHER-HOLE" OR "TUNNEL" METHOD OF BLASTING.

Another method, known in the East as "tunnel" blasting, in the Middle West as "gopher-hole" blasting, and on the Pacific coast as "coyote-hole" blasting, consists in firing large charges in small tunnels driven into the quarry face at the floor level. The method is simply snake-hole blasting on a large scale. The tunnel or drift is made as small as can be conveniently driven, being usually 3 to 4 feet wide and 4 feet high. The tunnel driving is usually done by contract at a price ranging from $1.50 to $2 per foot. Plate II, *C* (p. 40), shows the opening of a "gopher hole" at the base of a ledge. As a rule, the tunnel is driven 40 or 50 feet and then right and left cross headings are driven at right angles to the main leg, thus making a T-shaped opening. The dynamite is all placed in the cross headings and none in the main leg. It can be packed more closely if the cartridges are removed from the boxes. The charge should be divided into several parts having intervals of 15 to 20 feet between, filled with muck. Some quarrymen fill the intersection of the legs and at least half of the main leg with lean concrete, which is allowed to stand for at least 48 hours. Owing to the danger of premature discharge or damage by water seepage where a charge is left standing for 48 hours, a better practice is to build rough arches of small bowlders facing the charge in each branch and in the main leg. The charges should be wired in parallel and fired with a power current if possible. There should be at least two detonators in each explosive chamber, and to insure against misfires the chambers should be connected with trinitrotoluene detonating fuse.

The method is best adapted for quarrying a high face where the strata are irregular or conditions make it difficult to operate cable drills, and is not suited for working a face less than 70 or 80 feet high. The maximum height permissible is 150 to 175 feet. If the face is more than about 100 feet, it is wise to sink a few supplementary churn-drill holes from the surface between the back line of the tunnel and the face. These holes should be about one-third the depth of the face. When loaded with light charges and fired simultaneously with the main charge they assist greatly in breaking the upper rock.

EXAMPLES OF "GOPHER-HOLE" BLASTS.

The charge used in a New York quarry for a "gopher-hole" blast supplemented by blasts in churn-drill holes consisted of 13,700 pounds of 60 per cent gelatin dynamite and 15,350 pounds of granulated nitroglycerin blasting powder. The blast threw down about 72,000 cubic yards of stone, or approximately 5.6 tons per pound of explosive.

The charge for another blast of the same type consisted of 23,575 pounds of 60 per cent gelatin dynamite and 26,550 pounds of granulated nitroglycerin blasting powder. About 210,000 cubic yards of rock were moved, or about 9.5 tons per pound of explosive.

TYPES OF EXPLOSIVES USED.

The types of explosives commonly used in quarrying rock for cement manufacture are "straight" nitroglycerin dynamite, ammonia dynamite, gelatin dynamite, low-freezing dynamite, and nitrostarch blasting powder. A detailed discussion of the composition and properties of these explosives is given in Bulletin 48.[a]

Straight nitroglycerin dynamite was in past years commonly employed for primary shots, but recently has given place largely to ammonia dynamite and nitrostarch blasting powder. The compositions of typical explosives of the former type are shown in Table 4.

TABLE 4.—*Compositions of 40 per cent and 60 per cent "straight" nitroglycerin dynamites.*[b]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	40	60
Wood pulp	15	16
Sodium nitrate	44	23
Calcium or magnesium carbonates	1	1
	100	100

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[a] Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 8, Bureau of Mines, 1913, 50 pp.

[b] Hall, Clarence, and Howell, S. P., work cited, p. 7.

Ammonia dynamite is now commonly employed. Its tendency to absorb moisture condemns its use in wet quarries. The compositions of typical ammonia dynamites sold in the United States are shown in Table 5.

TABLE 5.—*Compositions of typical 40 per cent and 60 per cent ammonia dynamites.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	22	35
Ammonium nitrate	20	30
Sodium nitrate	42	24
Combustible material [b]	15	10
Calcium carbonate or zinc oxide	1	1
	100	100

[a] Hall, Clarence, and Howell, S. P., work cited, p. 8.
[b] Consisting of wood pulp, flour, and sulphur.

In quarries where much water is encountered gelatin dynamite has been used with success. Gelatin dynamite contains a small percentage of nitrocellulose, which gelatinizes the nitroglycerin and renders it impervious to water. The compositions of typical gelatin dynamites as sold in the United States are shown in Table 6.

TABLE 6.—*Compositions of typical 40 per cent and 60 per cent gelatin dynamites.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	33.0	50.0
Nitrocellulose	1.0	1.9
Sodium nitrate	52.0	38.1
Combustible material [b]	13.0	9.0
Calcium carbonate	1.0	1.0
	100.0	100.0

[a] Hall, Clarence, and Howell, S. P., work cited, p. 8.
[b] Wood pulp is used in 60 per cent strength gelatin dynamite. Sulphur, flour, wood pulp, and sometimes resin are used in other grades.

Low-freezing dynamites are recommended for blasting in winter. Their freezing point is about 35° Fahrenheit. Other types of explosives may be kept from freezing, or if frozen may be safely thawed by using proper apparatus,[a] but there is still danger that the explosive may become chilled from the air or from the cold rock while being loaded. Compositions of typical low-freezing dynamites are given in Table 7.

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[a] See Hall, Clarence, and Howell, S. P., Magazines and thaw houses for explosives: Tech Paper 18, Bureau of Mines, 1912, 34 pp.

TABLE 7.—*Compositions of typical 40 per cent and 60 per cent strength low-freezing dynamites.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	30	45
Nitrosubstitution compounds	10	15
Combustible material [b]	15	16
Sodium nitrate	44	23
Calcium or magnesium carbonate	1	1
	100	100

[a] Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering operations: Bull. 48, Bureau of Mines, 1914, p. 7.
[b] Composition similar to that of the "straight" nitroglycerin dynamites.

Nitrostarch blasting powders are used in a number of quarries. They are relatively insensitive to shock and are consequently safer to handle than more sensitive types. The compositions of typical nitrostarch blasting powders used in the United States are given in Table 8.

TABLE 8.—*Composition of typical 40 per cent and 60 per cent strength nitrostarch blasting powders.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitrostarch	40.0	46.0
Sodium nitrate	52.5	38.5
Ammonium nitrate	3.0	14.0
Sulphur	3.0	....
Oil	0.5	0.5
Chalk	1.0	1.0
	100.0	100.0

[a] Table supplied by S. R. Russell, of E. I. du Pont de Nemours & Co., Wilmington, Del.

Much useful information on the use of explosives in quarrying may be found in Bulletin 80 of the Bureau of Mines.[a]

RATE OF DETONATION.

An important feature of explosives, from the quarryman's viewpoint, is the rate of detonation. Explosives that have a relatively slow rate of detonation, such as the lower grades of nitroglycerin dynamite, tend to push out masses of rock rather than to shatter them, or, in other words, the effect is propulsive rather than disruptive. On the other hand, explosives that have a rapid rate of detonation, such as the higher grades of nitrogylcerin dynamite, tend to shatter the rock; that is, their effect is primarily disruptive.

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[a] Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen: Bull. 80, Bureau of Mines, 1915, 125 pp.

In blasting rock for the manufacture of cement, the aim of the quarryman is to break the rock into small pieces, hence the explosive used should have great disruptive force. "Straight" nitroglycerin and gelatin dynamites develop the strongest disruptive force of any of the ordinary explosives available for quarry work.

GASES EVOLVED BY EXPLOSIVES.

Carbon monoxide and hydrogen sulphide are the two most common poisonous gases formed by the detonation of explosives. Hydrogen sulphide can be easily recognized by its strong characteristic odor. Carbon monoxide is invisible and odorless and its effects are swift and deadly. It is, therefore, the most to be feared of all the gases evolved. Carbon monoxide is produced chiefly by the low-freezing and "straight" nitroglycerin dynamites, and hydrogen sulphide by the gelatin dynamites and black blasting powder.

Considerable quantities of the oxides of nitrogen may be formed when the explosive burns rather than detonates. This may occur from the side-spitting of a fuse or through the use of weak detonators. After careful determinations, Mann[a] found that where complete detonation occurs, very small amounts of the oxides of nitrogen are formed.

The fact should be emphasized that when detonation is incomplete, poisonous gases of various kinds are produced in excessive amount. Therefore, greater safety, as well as increased efficiency, is brought about by strong and complete detonation of the charge.

As most quarrying is of the open-pit type, the gases from small blasts can usually escape readily. Where large blasts are fired, however, especially in deep quarries, sometimes too little attention is given to the dangers from the gases evolved. An instance has been recorded[b] of the poisonous gases resulting from a heavy blast causing the death of 7 men and rendering 40 others unconscious. They entered the quarry about one-half hour after the blast had been fired. Obviously, after a large blast, sufficient time should be allowed for the gases to escape before anyone is permitted to enter the quarry. In any event enough time should be allowed to render the men safe from possible delayed explosions.

In a number of cement-manufacturing localities the rock is mined, and as the blasting is in tunnels the presence of poisonous gases may constitute a grave danger to the workmen. For underground work those explosives should be selected that will break the rock efficiently and at the same time evolve a minimum of poisonous gases. Through experiments conducted by the Bureau of Mines,[a] it was found that the amount of poisonous gases evolved could be decreased by reducing the proportion of combustible material in the explosive and in the paper wrapper containing it. Such a reduction has the effect of increasing the amount of oxygen available for combustion. The tests showed that the proportion of poisonous gases was lowest with gelatin dynamite and may be further reduced by diminishing the content of combustible material, and also by substituting flour for sulphur, wood pulp, or resin in the combustible portion of the explosive. It is recommended, therefore, that for underground blasting a "40 per cent" gelatin dynamite of the following composition be used:

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[a] Mann, E. A., Report on investigations into the compositions of the gases caused by blasting in mines: Perth, Australia, 1911, pp. 14–15.

[b] See Munroe, C. E., and Hall, Clarence, A primer on explosives for metal mines and quarrymen: Bull. 80, Bureau of Mines, 1915, p. 16.

Special formula for 40 per cent strength gelatin dynamite.[b]

Nitroglycerin	33
Nitrocellulose	1
Sodium nitrate	54
Combustible material[c]	11
Calcium carbonate	1
	100

Where blasting is conducted in confined drifts, the use of a jet of compressed air at the face assists in driving the poisonous gases away. Where work is carried on in several places simultaneously, it is wise for the men, after a blast has been fired, to work for a few hours at another breast and thus allow sufficient time for the gases to escape.

AMOUNT OF EXPLOSIVE FOR PRIMARY SHOTS.

In proportioning a charge, good judgment should be exercised. The object to be kept in view in blasting is to shatter the rock into reasonably small pieces and at the same time move it as little as possible.

There are several disadvantages in blasting in such a manner that the rock is thrown too far. The most obvious loss in overcharging is the waste of explosive involved. Also, if the rock is scattered far from the face, the cost of loading the scattered fragments is excessive where steam shovels are employed. The disadvantage of overcharging is, in this respect, most pronounced where the face is high.

Moreover, the tracks for cars or steam shovels are usually close to the quarry face. If the tracks are where rock may be thrown on them by the blast, considerable time and labor are required to move them, or, if left in position, to remove obstructions and repair damage caused by flying rock. Furthermore, the danger to the workmen is great where rock is likely to fly long distances.

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[a] See Hall, Clarence, and Howell, S. P., work cited, p. 10.

[b] Hall, Clarence, and Howell, S. P., work cited, p. 11.

[c] Flour.

GOOD JUDGMENT AND EXPERIENCE REQUIRED.

Lack of good judgment on the part of a blasting foreman may cause much trouble and annoyance to the men handling the rock. An instance was noted where, through poor judgment, the blasting foreman advised removal of the tracks from the vicinity of the quarry face. When the shot was fired the rock was moved so little that no damage or obstruction would have been caused to the tracks. For a subsequent shot the foreman expressed the opinion that the tracks were in no danger. When the shot was fired they were buried with rock.

Good judgment in determining the amount of explosive necessary is manifested also in the effect produced on the rock mass. When drill holes in a low bench are loaded too heavy, the blast tends to throw out a trench along the line of holes. When this takes place the rock near the holes is thrown too far and, at the same time, the blast loses much of its effectiveness for shattering the rock farther from the holes. The throwing out of a trench in this manner is known locally as a "chimney" shot. At one quarry visited where an overcharged blast was fired the charge was sufficient to blow out a trench nearly to the bottom of the quarry face, which at that point was 22 feet high. On the other hand, if the holes are loaded too light the rock is insufficiently shattered and much secondary blasting is required.

On a bench less than 30 or 35 feet high the ideal shot thoroughly shatters the mass and moves it but little from its original position. This result is best accomplished by employing the "buffer" method, as described in a subsequent paragraph. As a result of such a shot, the mass of shattered rock may, on account of the air spaces formed within it, stand up several feet higher than the face, and is in excellent condition for loading with steam shovels.

Success in loading drill holes depends on good judgment and experience. Experience gained in one locality may not serve for another locality on account of variations in local conditions. Each rock deposit has its own peculiar features, and it is only by long experience that one can become sufficiently familiar with the conditions to attain a high degree of efficiency in blasting.

CALCULATION OF CHARGE.

Mere guesswork in proportioning the charge can not be too strongly condemned. Many failures of shots are due to the crude methods of determining the amount of explosive used. When churn-drill methods are first tried, a common mistake is to make the burden too heavy. It is then impossible to place enough explosive in the drill holes to effectually shatter the mass of rock blocked out. Blasting in churn-drill holes, as practiced in many quarries at the present time, involves large costs for drilling and explosives for a single shot. Where the cost of one shot may reach several thousands of dollars, failure of the shot results in considerable loss. Success in blasting depends largely on the proper proportioning of the charge, consequently a careful and systematic method of determining the charge should be followed.

The best method is to calculate the tonnage of rock to be moved and to proportion the charge accordingly. The tonnage in short tons may be easily calculated by multiplying together the length, breadth, and height, in feet, of the mass to be moved, multiplying this product by the weight of a cubic foot of the rock, and dividing by 2,000. An average weight per cubic foot of solid limestone is 168 pounds. If a reliable determination has been made of the weight of the rock in any quarry, the figure obtained may be used in place of 168.

If, for example, the length of the mass of rock to be shot down is 200 feet, the width 25 feet, and the height 80 feet, the approximate tonnage would be $\frac{200 \times 25 \times 80 \times 168}{2000} = 33{,}600$ tons.

The amount of explosive required for a given tonnage depends on the physical properties of the rock, such as hardness, soundness, the attitude of the beds, and also on the type of explosive employed. As a rule, 1 pound of the explosive ordinarily used in churn-drill holes is sufficient for breaking 4 to 5 tons of rock. An exact figure can be determined only by experience. If the charge is calculated on a basis of 1 pound of 40 per cent "straight" nitroglycerin dynamite or nitrostarch blasting powder for each 5 tons of rock, and the rock is not sufficiently shattered, either the proportion of explosive should be increased or a higher grade should be employed. If, on the other hand, a V-shaped mass is blown out and the rock is hurled to a great distance, subsequent charges may be reduced in size or a weaker explosive may be substituted.

The amount of explosive as determined on some such basis may be proportioned among the various drill holes. Even when all the holes are of equal depth, it may be wise to vary the charges. In blasting a heavy toe, or a corner, heavier charges may be required for certain holes than for others.

Usually the drill holes are not of equal depth because the surface is curved, sloping, or irregular. Under such conditions it is necessary to calculate the charge for each drill hole. To simplify such determinations, Table 9 has been compiled, showing the number of tons of rock for each foot of drill hole for various spacings and burdens. The figures in the horizontal line at the top represent the number of feet of spacing, and in the column at the left, the number of feet of burden. Thus, by following to the right the line for any particular burden until the column under the desired spacing is reached, the figure found will represent the number of tons of rock that must be moved for each foot of drill hole. If, for example, the burden is 14

TABLE 9.—*Number of tons*[a] *of limestone per foot of drill hole for various spacings and burdens.*

Burden of drill holes.	Spacing of drill holes, feet.															
	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Feet.	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*
5	2.08	2.52	2.92	3.36	3.78	4.20	....	....	....	....	....	....	....	....	....	....
6	2.52	3.02	3.52	4.03	4.53	5.04	5.54	....	....	....	....	....	....	....	....	....
7	2.94	3.52	4.11	4.70	5.29	5.88	6.46	7.05	....	....	....	....	....	....	....	....
8	3.36	4.03	4.70	5.37	6.04	6.72	7.39	8.06	8.73	....	....	....	....	....	....	....
9	3.78	4.53	5.29	6.04	6.80	7.56	8.31	9.07	9.82	10.58	....	....	....	....	....	....
10	4.20	5.04	5.88	6.72	7.56	8.40	9.24	10.08	10.92	11.76	12.60	....	....	....	....	....
11	....	....	6.46	7.39	8.31	9.24	10.16	11.08	12.01	12.93	13.86	14.78	....	....	....	....
12	....	....	....	8.06	9.07	10.08	11.08	12.09	13.10	14.11	15.12	16.13	17.13	....	....	....
13	....	....	....	....	9.82	10.92	12.01	13.10	14.19	15.28	16.38	17.47	18.56	19.65	....	....
14	....	....	....	....	....	11.76	12.93	14.11	15.28	16.46	17.64	18.81	19.99	21.16	22.34	....
15	....	....	....	....	....	....	13.86	15.12	16.38	17.64	18.90	20.16	21.42	22.68	23.94	25.20
16	....	....	....	....	....	....	....	16.13	17.47	18.81	20.16	21.50	22.84	24.19	25.53	26.88
17	....	....	....	....	....	....	....	....	18.56	19.99	21.42	22.84	24.27	25.70	27.13	28.56
18	....	....	....	....	....	....	....	....	19.65	21.16	22.68	24.19	25.70	27.21	28.72	30.24
19	....	....	....	....	....	....	....	....	20.74	22.34	23.94	25.53	27.13	28.72	30.32	31.92
20	....	....	....	....	....	....	....	....	....	23.52	25.20	26.88	28.56	30.24	31.92	33.60
21	....	....	....	....	....	....	....	....	....	24.69	26.46	28.22	29.98	31.75	33.51	35.28
22	....	....	....	....	....	....	....	....	....	....	27.72	29.56	31.41	33.26	35.11	36.96
23	....	....	....	....	....	....	....	....	....	....	28.98	30.91	32.84	34.77	36.70	38.64
24	....	....	....	....	....	....	....	....	....	....	....	32.25	34.27	36.28	38.30	40.32
25	....	....	....	....	....	....	....	....	....	....	....	33.60	35.70	37.80	39.90	42.00
26	....	....	....	....	....	....	....	....	....	....	....	....	37.12	39.31	41.49	43.68
27	....	....	....	....	....	....	....	....	....	....	....	....	38.55	40.82	43.09	45.36
28	....	....	....	....	....	....	....	....	....	....	....	....	....	42.33	44.68	47.04
29	....	....	....	....	....	....	....	....	....	....	....	....	....	43.84	46.28	48.72
30	....	....	....	....	....	....	....	....	....	....	....	....	....	....	47.88	50.40

[a] Rock assumed to weigh 168 pounds per cubic foot.

feet and the spacing 11 feet, 12.93 tons of rock must be moved for each foot of drill hole. By multiplying this quantity by the depth of the hole in feet, the total tonnage for which explosive must be provided in this hole may be determined.

Obviously, the proper spacing and burden for a given size of drill holes may be determined in the same way. Thus it may be found that when the calculated amount of explosive is placed in a 5⅝-inch drill hole the charge is too low in the hole to break the upper rock effectually. The difficulty may be overcome by separating the charge into two parts with intermediate stemming, by using a larger amount of a lower grade explosive, by increasing the burden and spacing of the drill holes and increasing the charge accordingly, or by reducing the diameter of the drill holes. When the explosive would fill the bore hole too high, it may be a good plan to chamber the hole. If the drill holes will not hold enough 40 per cent strength explosive to properly break the bottom, it is better to use 60 per cent strength explosive in the bottoms of the holes.

In order to judge the proper size of drill holes to use, the shot firer should know the number of pounds of explosive that can be placed in a drill hole of a given size. Table 10, compiled by one explosives firm,[a] indicates the quantity of various types of explosives that can be placed in each foot of drill holes of various sizes when the cartridges are slit and the charge well tamped.

TABLE 10.—*Quantity of explosive per foot that can be placed in drill holes of various sizes.*

Diameter of drill hole.	Gelatin dynamite.	Straight nitro-glycerin dynamite.	Low freezing ammonia dynamite.	Extra ammonia dynamite.
Inches.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*
3	4.25	3.75	3.60	3.72
3½	5.68	5.10	4.89	5.07
4	7.55	6.60	6.33	6.62
4½	9.35	8.40	8.06	8.38
5	11.80	10.50	10.08	10.35
5⅝	14.94	13.20	12.53	13.00
6	17.00	15.00	14.40	14.90

The conditions of a shot may be varied by changing the amount of the charge, by changing the type of the explosive, and by varying the spacing and burden of the drill holes. Hence the problem is complicated, and good judgment and long experience are required on the part of the shot firer before maximum efficiency is attained.

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[a] See E. I. Du Pont de Nemours & Co. catalogue: Explosives for quarry blasting, 1915, p. 15.

"BLANKET" OR "BUFFER" BLASTING.

The effectiveness of blasting may, under certain conditions, be greatly facilitated by "blanket" or "buffer" blasting, also termed "shooting against the bank." This method can be employed only when the face does not exceed 30 or 35 feet in height. In loading the shattered rock with a steam shovel a strip 14 to 20 feet wide is left next to the face. This acts as a buffer for the next blast and prevents the rock from being hurled into the excavation where it may bury tracks, damage equipment, and increase the cost of loading. Also, the buffer tends to confine the energy of the explosion and promote shattering. In several Illinois quarries buffer blasts thoroughly shatter the rock with so little displacement that the casual observer would not realize that a blast had been discharged. Some quarrymen claim that blanket blasting is more successful with a good floor seam than where no seam exists. The method is highly recommended for quarries with low faces, but can not be employed with a high face, for the shattered rock falls to the floor and leaves the upper part of the face exposed. In loading holes for a buffer blast, the charge should not extend to the upper part of the drill hole, because the buffer tends to direct the shock of the explosion back into the solid ledge, and "breakbacks" may complicate the drilling of the next line of holes.

ARRANGEMENT OF CHARGES IN DRILL HOLES.

THE CONTINUOUS CHARGE.

A continuous charge of one type of explosive that fills the hole except for the space at the top required for stemming is the simplest arrangement possible. Such a charge may be used where the rock is uniformly hard, the face vertical, and the toe well cleaned. The burden, spacing, and diameter of drill holes should be adjusted to conform with the quantity of explosive as governed by the rock tonnage effectively moved per pound of explosive. The explosive used may be 40 per cent strength dynamite or of higher grade. In no quarry observed was a grade lower than 40 per cent strength used for a continuous charge. The use of "40 per cent" dynamite is common, but "60 per cent" dynamite is used in a number of quarries. It is claimed that the more effective shattering obtained with the stronger explosive more than justifies the higher cost. However, opinions differ regarding the success of using the higher explosive. Some quarrymen claim that "60 per cent" dynamite pulverizes the rock adjacent to the drill holes but does not shatter the rest of the burden more than "40 per cent" dynamite. The action of the explosive is probably controlled to some extent by the physical properties of the rock. In this, as in many other matters, definite rules can not be given. The quarryman must be guided to some extent by the characteristics of the rock, and the best methods can in most places be determined only by experience.

In general, the higher explosive breaks the rock into smaller fragments than that of lower grade. It must be remembered, however, that the results obtained depend on spacing of holes and size of charge, as well as on grade of explosive used. If the rock is extremely unsound—that is, intersected by numerous seams—a low-grade explosive will probably give the best results.

USE OF DIFFERENT GRADES OF EXPLOSIVE IN THE SAME DRILL HOLE.

The continuous charge is not always made up entirely of one type of explosive. In many quarries the burden is heavier near the quarry floor, consequently a greater volume of explosive, or a more powerful explosive, must be used at the bottom. If the hole is not sprung and the charge is continuous, the charge can not be concentrated in the bottom of the hole except by using a higher grade of explosive at that point. Under such conditions, "60 per cent" dynamite is commonly used in the bottom of the hole, and "40 per cent" dynamite for the rest of the charge. Occasionally as high a grade as "75 per cent" gelatin dynamite is used in the bottoms of drill holes where the lower beds are exceptionally hard.

Failure to employ sufficient explosive near the base of the quarry wall may later cause much delay and additional expense. An instance was observed where an unbroken bench about 8 feet in height was left at the quarry wall. This bench had to be cleaned, redrilled, and blasted before the cut could be completed.

The upper part of a shale or limestone deposit, especially in flat-lying rock, is usually thin bedded or jointed and is easier to break than the lower part. Consequently, a low-grade explosive, commonly of 30 per cent strength, is used for the upper part of the charge in such deposits.

In some quarries an effective means of breaking the rock is to throw out the base with a heavy charge and allow the upper part to fall, the impact of the fall assisting greatly in shattering the mass. Where this method is employed a high-grade explosive is usually placed in the bottom of the hole and a small amount of low-grade explosive in the upper part.

For rock having "slab cleavage," that is, rock in which the length and width of the average block is much greater than its thickness, this method is to be avoided. If the base is thrown outward and the upper beds fall almost vertically, the attitude of the fragments will be similar to that illustrated in *A*, figure 7. The slab-like masses may be so bound in that great difficulty will be encountered in loading them. Not only is loading rendered difficult; but the loading tends to undermine the pile of blocks, involving great danger from slides. This danger is especially pronounced in the wintertime. However, if the charge is so adjusted that the upper beds are thrown farthest from the face and the lower beds successively to lesser distances, the rock fragments are not bound in, but lie in the most favorable position for loading, as shown in *B*, figure 7.

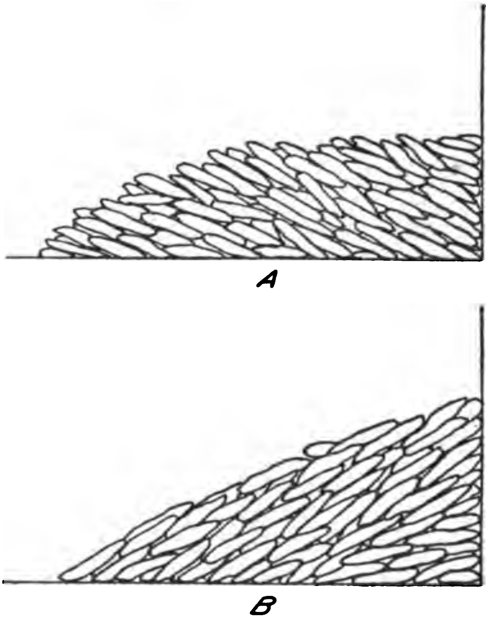

FIGURE 7.—Effects of different methods of placing charge in rock having "slab cleavage": *A*, Manner in which the slabs are bound in by blast that throws base farther than upper part; *B*, favorable position of rock from blast that throws upper part farther than lower part.

The use of different grades of explosive in the same drill hole is opposed by some quarrymen on the ground of lack of effectiveness. It is claimed that because the different grades of explosive have different rates of detonation the shot is not exactly simultaneous, the higher grade detonating an appreciable interval ahead of the lower grade, and that this lack of coordination between the different parts of the shot weakens the force of the explosion. This objection is probably not valid, for the detonation of each grade starts at the same moment, and as each grade is exerting its force on a different part of the rock mass it is probable that the time interval is too small to affect the result materially.

THE BROKEN CHARGE.

To obtain a successful blast, proper distribution of the charge in the drill hole is essential. To accomplish the desired distribution, the charge may have to be divided into two or more parts with intermediate stemming. Adjacent holes may be so loaded that explosive in one hole is at the same level as stemming in the next. The point should be emphasized, however, that where the rock is of uniform hardness, the same distribution may be attained by reducing the size of the drill hole and employing a continuous charge. The continuous charge should be used wherever possible.

In ledges where the successive beds of rock vary in soundness or hardness and therefore break with varying degrees of difficulty, proper distribution of the charge is best attained by the use of intermediate stemming. Where the rock varies widely in hardness, it is customary to put explosive in those parts of the holes that intersect the hard beds and stemming in the parts that intersect the softer beds. Where shale is interbedded with limestone, as a rule no explosive is used in that part of the drill hole that intersects the shale, because it is a soft rock and is easily broken. Explosive is therefore placed in the limestone and stemming in the shale.

The separation of a charge into five or six parts, with intermediate stemming, has been noted. If electric detonators are employed to fire the charge, a number of detonators are required for a charge of this type.

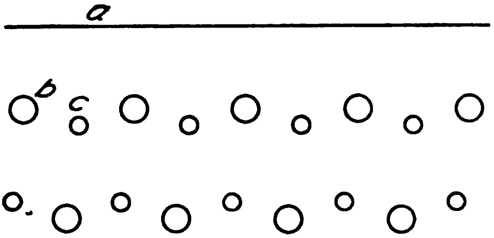

FIGURE 8.—Arrangement of drill holes in an Iowa quarry: *a*, Face; *b*, churn-drill holes; *c*, supplementary tripod-drill holes.

EXAMPLE OF A BROKEN CHARGE.

In one limestone quarry in Iowa the lower beds are hard and fairly sound; the middle beds have numerous closed seams that cause the rock to break readily; and near the surface is a bed 2 to 6 feet thick of extremely hard blue limestone. Two rows of holes 5½ inches in diameter are made with a churn drill, the holes being staggered, placed 10 feet apart, and having 10 feet of burden. The churn-drill holes are supplemented by tripod-drill holes that are drilled into the hard blue limestone, but no deeper. It is stated that if such supplementary charges were not provided, the blue limestone would break into masses 30 to 40 feet in length and weighing many tons. The arrangement of the holes is shown in figure 8. The distribution of the charge in the churn-drill holes is illustrated in figure 9. Enough "40 per cent" dynamite is placed in the hole to reach to the top of the sound bed. This requires about 60 pounds and, to insure against misfire, two electric detonators are placed in this charge. The middle part of the hole is then filled with stemming up to the hard blue

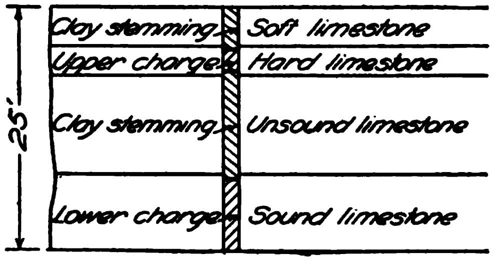

FIGURE 9.—Distribution of explosives in the churn-drill holes shown in figure 8.

bed. In the part penetrating this bed a second charge, consisting of 10 to 15 pounds of "40 per cent" dynamite, is placed. One electric detonator is used in this charge. The rest of the hole is tamped with clay stemming. This method is successful, especially since the introduction of the supplementary tripod-drill holes.

CHARGING HOLES WITH UNEQUAL BURDEN.

Rarely, if ever, are all the holes for one shot charged with the same amount of explosive, the charge being varied to suit conditions. A corner hole, for example, must be loaded heavier than the others to clean out the corner efficiently, or, as a quarryman would say, to "pull the corner." The amount of charge is also influenced by inequalities of the surface. A heavy toe or projections on the face require heavy charges in the holes immediately behind them. If on account of the presence of loose rock a hollow place in the face was not observed in time to modify the spacing of drill holes, the charge must be modified. The holes immediately behind the hollow place should be loaded lighter than the average, and those on either side a little heavier.

EFFECT OF WATER IN DRILL HOLES.

As drill holes are usually completed a considerable time before loading, they are as a general rule partly filled with accumulated water. In some places the explosive is loaded into the holes without removing the water. If a type of explosive designed for submarine blasting is employed, this procedure may be permissible. Usually the use of other types of explosives is desirable, hence removal of the water is advisable. An objection to the presence of water, aside from the possible injurious effect on the explosive, is the resulting decrease in the efficiency of the explosion. It is a well-known law of physics that when the temperature of a gas in a sealed vessel is increased, the tendency of the gas to expand causes a corresponding increase in the pressure on the walls of the vessel. Therefore, the force of an explosion is due in large measure to the high temperature of the gases evolved. Water in the drill hole keeps down the temperature of the gases and reduces their pressure, and therefore detracts from the efficiency of the blast.

REMOVAL OF WATER FROM DRILL HOLES.

All quarrymen are familiar with apparatus for the removal of water from drill holes. A portable hoist operated by a gasoline engine or by electricity is convenient for bailing out deep holes. With the ordinary apparatus employed all the water can not be removed, but the small amount left in the bottom of the hole usually occasions no disadvantage.

LOADING DRILL HOLES.

It is an established fact that the explosive should completely fill that part of the drill hole that it occupies. Even a small air space around the charge greatly weakens the force of the explosion. A common practice is to cut the cartridge and pour the loose dynamite into the holes through a large funnel. This is somewhat dangerous, as loose dynamite may be scattered around the holes and cause a premature explosion. Steps are being taken in one State to make loading of loose dynamite illegal. However, in loading irregular holes in unsound rock the loose charge is preferred by some operators. When cartridges are used they should be large enough to fit the drill hole without danger of jamming, and may be slit in such a manner that they may be easily expanded to fill the hole when pressed with a wooden tamping rod. The cartridges that contain the detonators should not be tamped.

STEMMING AND TAMPING.

It is assumed by some quarrymen that explosives like 40 per cent "straight" nitroglycerin dynamite or of higher grade act so quickly that no stemming is necessary. Experiments conducted by the Bureau of Mines[a] have shown conclusively, however, that the efficiency of 40 per cent "straight" nitroglycerin dynamite is greatly increased by the use of stemming.

Clay is the most common stemming employed, although sand or rock dust is sometimes used. Experiments conducted by the Bureau of Mines[b] show that tamped moist fire clay gave the best results with 40 per cent "straight" nitroglycerin dynamite in four out of six tests with varying amounts of stemming, while tamped moist sand stood second. In the other two tests this order was reversed.

Greenwell and Elsden[c] claim that untamped dry sand is the best stemming. The advantages of sand stemming are thus enumerated:

1. The time taken in loading is reduced to about one-fourth.
2. The effect of the shot is at least as good as with clay stemming.
3. It has been found that with sand stemming rock fragments are not thrown as far as with clay stemming.
4. Misfires are not as common as with clay stemming, as no tamping rod is used and therefore no injury to the fuse or wires can arise.
5. If a misfire should occur, the stemming may be readily removed with a wooden implement without danger.
6. Premature explosions during tamping are impossible when sand is used.

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[a] See Snelling, W. O., and Hall, Clarence. The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1912, 20 pp.

[b] Snelling, W. O., and Hall, Clarence, work cited, p. 19.

[c] Greenwell, Allan, and Elsden, J. V., Practical stone quarrying, 1913, p. 287–288.

7. Sand is usually easily obtained.

8. Sand is more convenient than clay for making shots in open joints.

9. Dry sand does not freeze like clay.

It is also stated that the quarry-industry association of Germany investigated the use of sand stemming, and after numerous experiments the above conclusions were confirmed.

A number of the advantages enumerated are obvious, but some of the claims may be questioned. For example, the statement that sand is as effective as clay is not borne out by the Bureau of Mines tests. According to the tests made by Snelling and Hall, previously referred to, untamped dry sand was much less effective than either tamped moist sand or tamped moist fire clay, particularly where the larger amounts of stemming were employed. However, in the tests made by Snelling and Hall a perfectly smooth bore hole in metal was employed. A drill hole in rock is less smooth and regular, and consequently in actual practice different results may follow.

The claim that with sand stemming rock fragments are not thrown as far as when clay stemming is used is doubtful.

Sand stemming has without doubt many advantages in its favor and its use is recommended.

For blasting in churn-drill holes 35 to 90 feet deep, the depth of stemming used varies from 10 to 25 feet. In one cement-rock quarry in Pennsylvania where 50 per cent nitrostarch blasting powder is used, an average of 22 feet of clay stemming is employed in holes averaging 85 feet deep.

METHODS OF TAMPING.

For holes of moderate depth, tamping may be done with a light wooden pole. A metal tamping bar should never be used.

Attachment of a tamping block to a drill bar and tamping with a drill is extremely dangerous and should never be permitted. On January 10, 1907, a premature explosion occurred at the site of the Pedro Miguel Locks on the Isthmus of Panama. Ten men were killed. As a result of an investigation by the Isthmian Canal Commission it was decided that this and other premature explosions had been caused by the use of tripod rammers, and their further use was prohibited.

For deep holes a tamping block 3 or 4 feet long may be attached to a rope. A convenient device in tamping deep holes is a light tripod with a pulley at the apex over which the rope attached to the tamping block may be operated. The use of a tripod not only makes tamping easier, but also permits the men to work at one side and not directly over the loaded hole. This lessens the danger of accident in case of premature explosion.

GENERAL PRINCIPLES FOR THE SHOT FIRER'S GUIDANCE.

As a rule, a high explosive shatters rock more effectually than a low explosive; consequently, in quarries where many large solid blocks are thrown down by the primary shot, the substitution of a higher grade explosive may give more satisfactory results. However, the use of high-grade explosives is not recommended, as they are expensive and more dangerous to handle. The use of a detonating fuse increases the effectiveness of low-grade explosives and thus permits their use where otherwise a high-grade explosive is required. A variation in the arrangement of drill holes may bring about the same result as changing the type of explosive, for it has been found that, within certain limits, a reduction in the burden and spacing of drill holes causes more thorough shattering of the rock mass.

Drill holes should be of the minimum diameter required to contain the necessary explosive. Intermediate stemming is advisable in certain deposits where enlargements occur in the drill holes, where the beds are variable in soundness, or where for any other reason an adjustment of the powder column seems to be justified. However, if the successive beds are fairly uniform, equally efficient results may be attained by reducing the diameter of the drill holes and using a continuous charge. By this means the charge is distributed throughout a greater length of hole and the inconvenience of employing intermediate stemming is avoided.

Water should be removed from drill holes before loading.

If there is a heavy toe or a poor floor seam it is better to place in the bottoms of the drill holes explosive of higher grade than the main charge.

DETONATING THE CHARGE.

METHODS OF FIRING CHARGES.

In order to obtain the most effective results, a number of holes should be fired simultaneously. As it is physically impossible to fire charges simultaneously with ordinary fuses, electric detonators and an electric current are required. One method of firing charges is to place one or more electric detonators in the dynamite of each hole, connect them all with wires, and fire with an electric current. Another method is to use a detonating fuse in each hole and to fire this fuse at the surface. The fuse itself detonates at a very high rate and fires all the charges.

ELECTRIC DETONATORS.

An electric detonator, also called an electric exploder, is a blasting cap containing a high explosive, such as mercury fulminate, which is ignited by means of a wire through which the current of electricity passes. This wire is of small diameter and has such a high resistance that it is heated to incandescence, or fused, and thereby fires the priming. Electric detonators are made in various sizes. For ordinary blasts in quarries, No. 6 detonators are in common use. However, it has been proved in many quarries that the efficiency of a blast largely depends on the intensity of the initial explosion, and for this reason the use of No. 8 detonators is recommended. A more complete discussion of electric detonators is given by Hall and Howell.[a]

NUMBER OF DETONATORS FOR EACH CHARGE.

For small blasts in tripod or hammer drill holes one detonator is commonly used for each hole. While this is probably sufficient to fire the charge effectively, the danger of misfires is greatly reduced by using two detonators and wiring them in parallel for live-wire current and in series if a battery is used.

For large blasts in churn-drill holes two or more detonators are commonly used. In one Pennsylvania quarry, where 90-foot holes 5 inches in diameter are loaded with continuous charges of 40 per cent "straight" nitroglycerin dynamite, three No. 6 detonators are used in each hole, one near the bottom, one at the middle, and one near the top. They are wired in parallel and fired with the quarry current.

When broken charges with intermediate stemming are employed, each part of the charge should have at least one detonator, and to insure against misfires two detonators are preferable.

In one quarry in the South it is customary to place several unattached No. 6 detonators in the charge to act as "boosters" for the purpose of increasing the rate of detonation and thereby increasing the effectiveness of the shot. This practice is very dangerous, for a loose cap may lodge in a cavity or on a ledge on the drill-hole wall and may later be fired with the tamping rod, causing a premature explosion.

The results of tests made at the Pittsburgh station of the Bureau of Mines to determine the efficiency of such a method, described in Bulletin 59,[b] indicate that "extra detonators placed at 5 inches apart in a cartridge file of an insensitive explosive 40 inches long have a slight tendency to increase the propagation of the explosion wave, but that extra detonators placed 10 inches apart offer no advantages." It is pointed out that the explosion of the dynamite surrounding the unattached detonators may precede the explosion of the detonator and that the latter would, under such circumstances, explode in the products of combustion and therefore offer little if any advantage in extending or increasing the rate of the explosion wave. Hence, the distribution of unattached detonators throughout the charge probably offers little if any advantage.

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[a] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, Bureau of Mines, 1913, 73 pp.

[b] See Hall, Clarence, and Howell, S. P., work cited, pp. 68–69.

However, the presence of several electric detonators distributed throughout the charge and connected by wires offers a decided advantage. The detonators are all fired simultaneously and tend to make the detonation of all parts of the charge practically simultaneous, thus producing the maximum shattering effect.

The best type of "booster" is a brass tube 4 to 6 inches in length, containing trinitrotoluene detonating fuse with an electric detonator attached. It increases the rate of detonation and insures more complete detonation of a low-grade explosive. The advantage of using "boosters" with 40 per cent or 60 per cent "straight" nitroglycerin dynamite may be questioned.

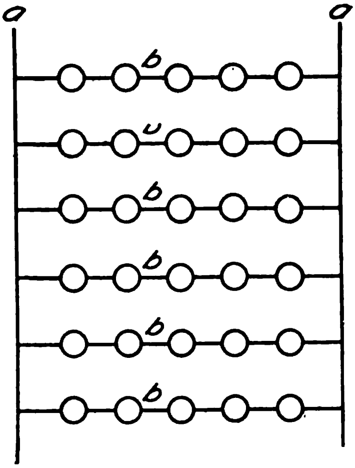

FIGURE 10.—Wiring in parallel series: *a*, Trunk lines; *b*, parallel series.

METHODS OF WIRING.

Two methods of wiring a number of shots to be fired at one time are in common use—wiring in "series" and wiring in "parallel." The methods are fully described in Technical Paper 111.[a] A third method, which is sometimes employed when many holes are fired at one time, is a combination of the above methods, and may be termed a "multiple series" or "parallel series" connection. When shots are fired in parallel a current of one ampere per detonator is recommended; therefore the use of a multiple connection for wiring a large number of holes would require a greater amperage than is usually available, and also would increase the difficulty of supplying suitable wiring. Connecting a large number of holes in series would require a high voltage and would involve the danger of a misfire in the event that insufficient voltage were available. Therefore, a method recommended is to connect small groups of holes in series and to connect these series in parallel, as shown in figure 10. When holes are connected in this manner it is important that each subseries should have the same resistance.

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[a] Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, pp. 21–22.

THE USE OF POWERFUL DETONATORS RECOMMENDED.

A point that requires special emphasis among quarrymen is the advisability of using strong detonators. As stated by Rutledge,[a] "when high explosives are detonated the stronger and quicker the action of the detonator the greater will be the shock given the explosive, and the more effective will be the explosive." A weak detonator may fire the charge, but may cause a very inefficient explosion, particularly when the more insensitive explosives are employed. The use of weak detonators simply because they are cheaper than strong ones is not wise. An imperfect detonation will probably result in a loss amounting to many times the price of all the detonators employed. No detonators weaker than No. 6 should be used with high explosives, and with the more insensitive explosives, such as gelatin or ammonia dynamite, or nitrostarch blasting powder, No. 8 detonators should be used. One blasting engineer advises the use of No. 8 detonators for all shots.

NUMBER AND ARRANGEMENT OF DETONATORS IN CHARGE.

Manufacturers of electric detonators claim that the priming exerts its greatest force in the direction of the loaded end. Consequently, the detonator shell should be so placed in the charge that its loaded end is toward the greater mass of explosive to be detonated. In any event the long axis of the shell should parallel the drill hole, and it should be entirely embedded in the explosive.

As a safety precaution, not less than two detonators should be placed in each charge. If a broken charge is employed, each separate part of the charge should have at least two detonators. Where heavy charges are fired in churn-drill holes, four or five detonators are commonly used in each hole, being distributed regularly throughout its length. Enough detonators should be used to insure sudden and complete detonation of the charge. Detonators cost only a few cents each, and the use of too few or too weak detonators may mean the loss of many dollars, and cause an accident if part of the charge is unexploded.

Each electric detonator should be tested with a reliable galvanometer. If any detonator shows abnormally high resistance, it should not be used when firing with a battery or where connected in series. The entire circuit should be tested in the same way when all connections are made except to the blasting machine or power circuit. In testing a circuit of holes with a galvanometer the instrument should have long lead wires and should always be placed at a safe distance. Ordinarily the current used in a galvanometer is much too weak to fire an electric detonator, but some galvanometers are operated with a shunt circuit, and a defect in wiring or connection may throw the entire current from a cell battery through the shunt. Such a current has been known to fire an electric detonator. Consequently, for the sake of safety, the lead wires should be long.

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[a] Rutledge, J. J., The use and misuse of explosives in coal mining: Miners' Circular 7, Bureau of Mines, 1914, p. 12.

For parallel connection there should be one ampere of current for each detonator, and for series or parallel series, at least two amperes for each series. When a blasting machine is used, the charges should always be connected in series.

The provision of ample current is important. Using a current much stronger than is required to insure against misfires has other advantages. Experiments by the Bureau of Mines[a] have shown that, owing to slight variations in electric detonators, those employed in a multiple shot do not all detonate at exactly the same moment; also, that when a weak current is used the time intervals between the detonations are longer than when a strong current is employed. Therefore, with a strong current the discharge of the various detonators is more nearly simultaneous, and consequently the blast is more effective.

DETONATING FUSE.

NATURE OF FUSE.

The type of detonating fuse known to the trade as "cordeau" is now in common use. It consists of a lead tube filled with trinitrotoluene and carefully drawn to uniform size. The diameter of the fuse commonly used is 6 mm., or $\frac{15}{64}$ inch. Also, 5.5 mm. size is used. Cordeau has been known for a number of years in France and England, but until recently it has been used in the United States only as a means of measuring the rate of detonation of high explosives.[b] During the last three or four years, however, cordeau has been used rather extensively in mining, quarrying, or other blasting work where large multiple shots are employed. It has been found especially advantageous in quarries where great masses of rock are thrown down with large charges of explosives in churn-drill holes.

METHOD OF USING DETONATING FUSE.

After the drill holes have been cleaned, and before loading is begun, a piece of detonating fuse is inserted into each hole. The fuse extends the full depth of the hole and 6 to 10 inches above the surface. After the holes are loaded and tamped, a main line of fuse is placed on the surface and attached to the branches from each hole. A detonator is attached to the end of the main line, and is fired either by fuse or by electric current, thus firing the detonating fuse. The detonation wave flashes along the main line and into each drill hole. As the rate of detonation of the trinitrotoluene is very high, all the charges are fired practically at the same moment. The method of placing the fuse is shown in figure 11.

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[a] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, 1913, p. 71.

[b] See Hall, Clarence, and Howell, S. P., work cited, pp. 66–68.

EFFICIENCY IN THE USE OF DETONATING FUSE.

It is claimed by Hall[a] that the use of detonating fuse is not as advantageous in the United States as in Europe, owing to the fact that the explosives used in the United States are more sensitive to detonation than those used in European quarries and mines. Explosives of the ammonia nitrate class used in the European countries are insensitive, and when detonators are used incomplete detonation may occur, whereas with cordeau detonation is more sudden and complete. In the United States, where nitroglycerin dynamites are commonly used, complete detonation may be easily brought about with electric detonators.

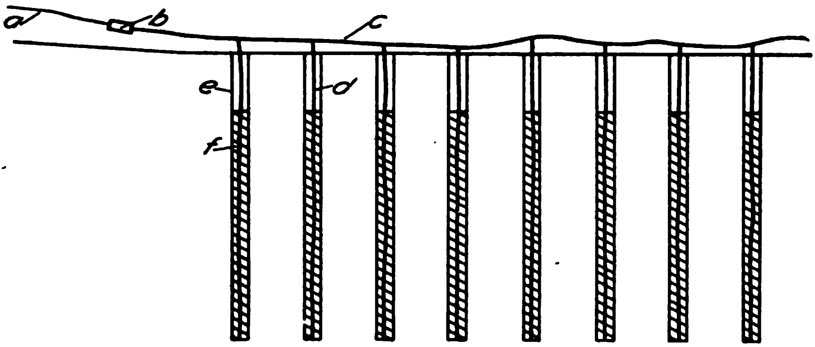

FIGURE 11.—Arrangement of detonating fuse in a circuit of holes: *a*, Wire; *b*, electric detonator; *c*, main line of fuse; *d*, branch line of fuse; *e*, stemming; *f*, charge of explosive in drill hole.

Also, the fact is generally recognized that for small shots in tripod or hammer drill holes the use of cordeau offers little if any advantage.

However, for deep-hole blasting, even where dynamite of the nitroglycerin class is used, detonating fuse has a number of advantages over electric detonators. Recognition of this fact has led to its recent introduction at a number of quarries in various parts of the United States. The different advantages may be summarized as follows:

Cordeau of 6-mm. diameter has a rate of detonation upwards of 15,000 feet a second. On this account the detonation of all the fuse used for a blast is practically instantaneous, and as fuse is in contact with each charge throughout the entire vertical depth it fires every part of the charge practically at the same moment. Not only is incomplete detonation impossible, but the simultaneous discharge increases the disruptive effect of the explosion. In general, therefore, for deep-hole blasting a smaller charge with cordeau will give as good results as a larger charge with electric detonators.

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[a] Hall, Clarence, Discussion of paper, "A new safety detonating fuse," by Harrison Souder: Am. Inst. Min. Eng. Bull., April, 1915, p. 895.

The rate of detonation of the fuse as compared with the rates of detonation of certain high explosives is given in the following table.[a]

TABLE 11.—*Rates of detonation of trinitrotoluene detonating fuse and of certain samples of high explosives.*

Class and grade of explosive.	Diameter of explosives tested.		Rate of detonation, feet per second.	Authority.
	Inches.	Millimeters.		
Trinitrotoluene detonating fuse	$\frac{10}{64}$	4	13,750	Bureau of Mines.[a]
Do	$\frac{15}{64}$	6	15,170	Do.
Do			17,000	Manufacturer.
60 per cent "straight" nitroglycerin dynamite	$1\frac{1}{4}$	32	20,490	Bureau of Mines.
50 per cent "straight" nitroglycerin dynamite	$1\frac{1}{4}$	32	18,850	Do.
60 per cent strength low-freezing dynamite	$1\frac{1}{4}$	32	16,640	Do.
40 per cent strength gelatin dynamite	$1\frac{1}{4}$	32	16,210	Do.
40 per cent "straight" nitroglycerin dynamite	$1\frac{1}{4}$	32	15,650	Do.
30 per cent "straight" nitroglycerin dynamite	$1\frac{1}{4}$	32	14,920	Do.
40 per cent strength ammonia dynamite	$1\frac{1}{4}$	32	10,350	Do.
5 per cent granulated nitroglycerin powder	$1\frac{1}{2}$	38	3,340	Do.

[a] Data for Bureau of Mines tests are from Bulletin 48, The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell, Bureau of Mines, 1914, p. 44.

It may be noted that the rate of detonation of the fuse is much higher than that of the 40 per cent strength ammonia dynamite, and about the same or a little lower than that of the 40 per cent "straight" nitroglycerin dynamite. One important advantage of the detonating fuse is that it increases the rate of detonation of any low-grade high explosive, for example, an explosive of the nitroglycerin class having a low content of nitroglycerin. Howell[b] has shown that the rate of detonation of a low-grade high explosive is increased to the rate of the detonating fuse extended through it. The disruptive effect of the low-grade explosive is thereby increased, but is not increased proportionally; that is, the disruptive force of a low-grade explosive detonated with cordeau is not as great as that of a similar type of explosive having a normal rate of detonation equal to that of cordeau. However, the disruptive force of the low-grade explosive is materially increased. Thus it may be seen that the quarryman may substitute low-grade and relatively cheap explosives for the higher grades ordinarily used and still maintain normal efficiency.

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[a] Howell, S. P., Discussion of paper "A new safety detonating fuse," by Harrison Souder: Am. Inst. Min. Eng. Bull. 100, April, 1915, p. 897.

[b] Howell, S. P., place cited.

It is interesting to note, however, that even when 40 per cent "straight" nitroglycerin dynamite or other types of higher grade are used, better results are attained by using cordeau than by electric detonators.

In one Indiana quarry the writer observed the result of a blast where cordeau had been used for the first time in this quarry. The explosive employed was 40 per cent nitrostarch blasting powder. The charge was approximately as large as was commonly used with electric detonators. The effect of the shot was to blow out a deep trench and to throw the rock so far that it buried tracks in the quarry. Evidently the charge was too heavy.

The writer observed a subsequent blast for which the charge was reduced about 25 per cent. For this blast 41 holes 19½ feet deep were drilled, the holes being arranged in two rows and staggered. The first row had about 10 feet of burden, and the holes were 12 feet apart. The second row was about 18 feet from the face, and the holes were 12 feet apart. An average of 45 pounds of "40 per cent" nitrostarch blasting powder was placed in each hole. For the blast referred to in the previous paragraph, 60 pounds was the average charge for each hole. The holes were tamped with clay and the shot discharged with cordeau fired by a single No. 6 detonator connected with the quarry current. The explosion shattered but did not throw the rock. A considerable number of blocks were large, requiring excessive secondary blasting. In the writer's opinion the size of the charge was reduced too much; a reduction of 10 to 15 per cent, rather than 25 per cent, in the amount of explosive employed would undoubtedly have given better results. The necessary reduction in the amount of the charge as compared with that employed with detonators is an excellent demonstration of the increased efficiency that results from the use of detonating fuse with an insensitive explosive.

In addition to the gain in shattering efficiency, it is claimed that the time required for loading holes is greatly reduced by the use of the detonating fuse.

SAFETY IN USE OF DETONATING FUSE.

The detonating fuse is much safer than detonators in storing or handling. Howell[a] states that trinitrotoluene is much less sensitive than either 40 per cent "straight" nitroglycerin or mercury fulminate. Trinitrotoluene can not be exploded by friction, fire, or ordinary shock, but requires the shock of a detonator. The fuse is not affected by heat or cold, but may be affected by moisture at an open end or where connections have been made.

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[a] Howell, S. P., work cited, p. 896.

The increased safety from the use of cordeau can best be appreciated by briefly reviewing the dangers that beset the ordinary method of firing a charge with electric detonators.

A detonator is usually placed at a point not far from the bottom of the drill hole. During all subsequent loading and tamping this detonator and others which may be successively placed in the hole are a menace to life and limb. Careless handling of the detonators or heavy, careless tamping of the charge may cause a premature explosion. In the event of a misfire, the presence of unexploded detonators along with dynamite in the rock mass is a source of extreme danger, as a blow from a hand sledge or from the tooth of a steam-shovel dipper may result in a serious or fatal accident. Occasionally "blind" misfires may occur; that is, one or more holes may not detonate, and, on account of the great mass of rock thrown down, this fact may be hidden from the quarryman. If the men are unaware of this danger no precautions may be taken and the risk will be greater than where misfires are known to have occurred. Incomplete detonation of a charge is also a source of danger. Unexploded dynamite in the rock, although less dangerous when no detonators are present, is a source of grave danger where metal tools are constantly employed.

When detonating fuse is used the drill holes contain no detonators, and the one used to fire the fuse need not be brought until everything else is prepared, the placing of the detonator being the last act in preparing a shot. Should a misfire occur, the lesser danger of the unexploded dynamite would be present, but the greater danger of unexploded detonators would be removed. However, misfires are rare when cordeau is used. They can occur only through serious defects in manufacture of the fuse, defective connections, or severance of the line in tamping. If due care is exercised, the danger of misfires is remote. "Blind" misfires can scarcely occur when cordeau is used, for the presence of the unexploded fuse could readily be observed.

When detonators are used misfires may be caused by attempting to fire so many holes that the firing machine is overloaded. With cordeau this danger is eliminated, for with proper connections an unlimited number of charges may be fired with one or two detonators.

Furthermore, the more efficient results from the use of detonating fuse permits in some places of the substitution of low-grade or insensitive explosives for those of higher grade. Such substitution decreases the danger, for it is well known that low-grade or insensitive dynamite is much safer to handle than high-grade dynamites, especially those of the "straight" nitroglycerin type.

PRECAUTIONS TO BE TAKEN IN USING DETONATING FUSE.

In placing detonating fuse in deep holes there is danger of the fuse breaking from its own weight. As pointed out by Howell,[a] the 6-millimeter fuse weighs about 10 pounds per 100 feet. Experiment showed that a piece of fuse 1 foot long would not support a weight of 34 pounds. Hence jerking or otherwise carelessly handling the fuse might easily break it. Such fuse is now prepared with a covering of fabric which greatly increases its strength. It is wise, therefore, to use covered fuse for deep holes.

A second danger is in tamping. The stemming employed should contain no sharp flakes or angular fragments of rock which might, when tamped, separate the fuse. Some misfires are presumed to have been caused in this way. Careless handling of the tamping stick itself might damage the fuse. The fuse protected with cotton covering is less liable to damage in tamping.

It is important that connections between the main line and the branches be properly made. The end that projects from the drill hole should be split for a distance of several inches and the main line placed in the crotch of the split. The two halves are then twisted about the main line in opposite directions, as shown in figure 12. It is important that the branch lines should be split and not the main line. Tests by the Bureau of Mines[b] have proved that to detonate cordeau through the lead covering is very difficult. With the method described it may be seen that the lead-covered fuse is the conductor of the detonation wave, and when it explodes, the branches, which are split in such a manner as to expose the explosive within, are easily fired. The union may be wound with tape to make it more secure and to keep out moisture. The branch line should be conducted away from the main line at right angles, at least for an inch or two, but at greater distances curves cause no difficulty. After all joints are made they should be carefully inspected before an attempt is made to fire the charge.

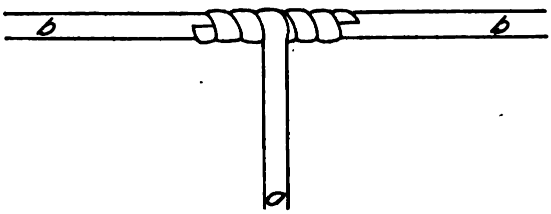

FIGURE 12.—Proper method of connecting branch lines and main line of detonating fuse: *a*, Branch line; *b*, main line.

Another method of making connections with the branch lines consists in eliminating the main line of cordeau entirely. A detonator is placed on the end of the fuse extending from each hole. Then the detonators are wired in series and fired with an electric current. By this method each hole is an independent unit, and the danger of misfires is thereby increased.

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[a] Howell, S. P., work cited, p. 898.

[b] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, Bureau of Mines, 1913, pp. 22–24.

The method of attaching a detonator to the detonating fuse is shown in figure 13. If the detonator is to be fired with ordinary fuse it must be crimped to the fuse in the ordinary manner. Usually, however, electric detonators are employed. The use of a No. 8 detonator is recommended. The first step in making the attachment is to cut off the end of the detonating fuse squarely. A brass union, which is slit at one end, is then slipped over the fuse, with the slit end projecting. The union is crimped to the fuse at the end opposite the slit. The detonator is then inserted into the union until it comes firmly against the end of the fuse. A space of one-eighth of an inch between the detonator and the fuse may be sufficient to cause a misfire. A ring is then slipped down over the union, and as the slits allow the union to close the detonator is held firmly in place.

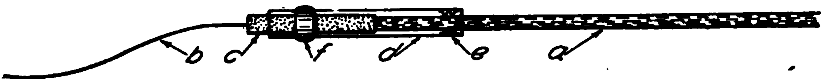

FIGURE 13.—Method of attaching electric detonator to detonating fuse: *a*, Fuse; *b*, wire; *c*, detonator; *d*, brass union; *e*, crimp in union; *f*, brass ring. After Souder.

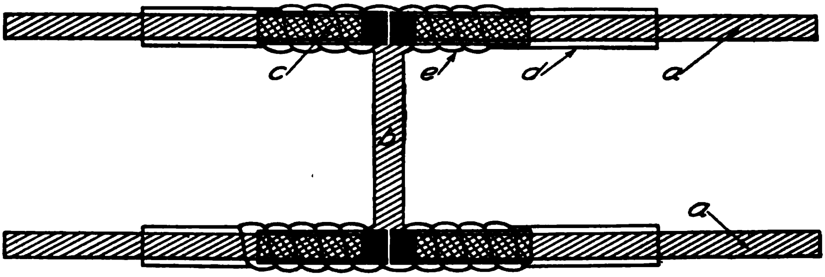

FIGURE 14.—Method of making cross connection with detonating fuse: *a*, Main lines of fuse; *b*, connecting branch of fuse; *c*, electric detonator; *d*, brass sleeve; *e*, winding of branch about main line.

If many holes are fired at one time, requiring two or more main lines, as a matter of precaution it may be advisable to make cross connections between the main lines. A method of making cross connections is described by Souder.[a] The main line is cut and electric detonators placed on the ends, which are then united by a brass sleeve firmly crimped to the fuse. The connecting fuse is then split and wound round the brass sleeve in the ordinary manner, as shown in figure 14.

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[a] Souder, Harrison, A new safety detonating fuse: Am. Inst. Min. Eng., Bull. 94, October, 1914, p. 2555.

The arrangement of detonating fuse and method of wiring for a blast in six rows of holes in a Pennsylvania quarry is shown in figure 15. To insure against misfire two detonators are employed, and each loop of fuse is fired at both ends.

RELATIVE COST OF FUSE AND DETONATORS.

A price quoted for cordeau in 1916 was $57.50 per 1,000 feet, bare, and $62.50 per 1,000 feet, covered. For a 20-hole shot, with the holes 50 feet deep and arranged in two rows with 12-foot burden and 14-foot spacing, the total cost of cordeau, including a surface main line, would be about $78. Two electric detonators for each hole and the necessary wiring would cost approximately $20. Though cordeau is more expensive than detonators in firing a charge, the increased efficiency with cordeau permits a compensating decrease in the size of the charge, particularly when gelatin or ammonia dynamite or nitrostarch blasting powder are used. The use of a detonating fuse for deep-hole blasting is to be recommended on the basis of economy and of safety.

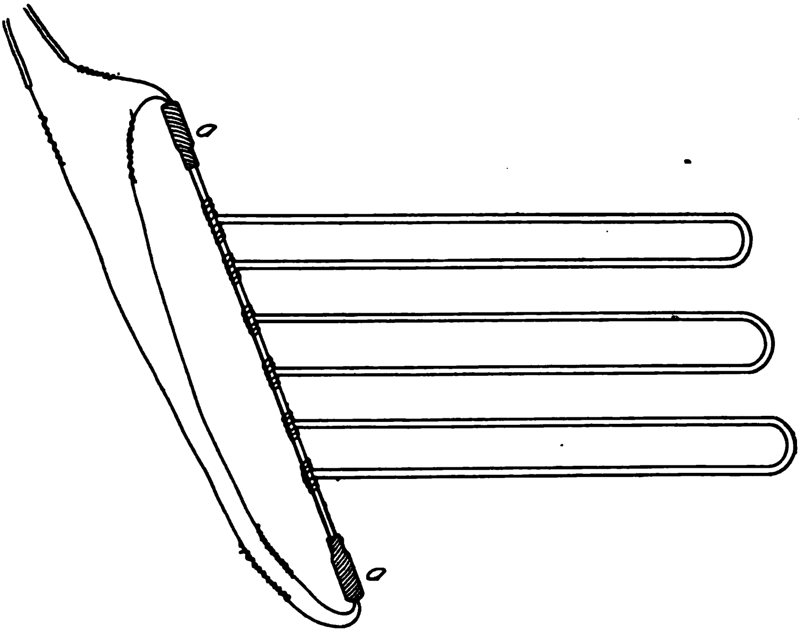


FIGURE 15.—Arrangement of surface lines of detonating fuse and wiring for a blast in a Pennsylvania quarry. Six rows of holes; double lines are detonating fuse; single lines are wires; *a a* are electric detonators.

EXAMPLES OF SHOTS IN WHICH DETONATING FUSE WAS EMPLOYED.

The details of a shot observed by the writer in a quarry near Bath, Pa., are given below.

The shot was in cement rock—that is, argillaceous limestone. The charge was distributed in 20 holes averaging 85 feet deep, the average depth of clay stemming used being 22 feet. "Fifty per cent" nitrostarch blasting powder was used. As shown in figure 16, the holes were arranged in two rows, staggered, and spaced 18 feet apart, the first row being 18 feet and the second row 36 feet from the face. The depth of the holes, depth of stemming, and number of pounds of explosive in each hole is shown in Table 12.

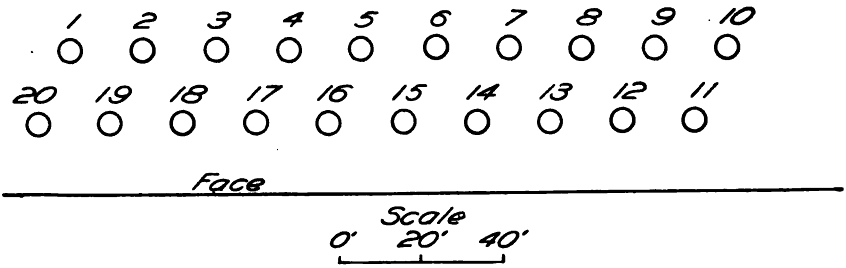


FIGURE 16.—Arrangement of drill holes for a blast in a quarry near Bath, Pa.

TABLE 12.—*Depth of drill holes, depth of stemming, and number of pounds of dynamite used for a typical blast in cement rock.*

Number of drill hole.	Depth of drill hole.	Depth of stemming.	Dynamite used.
	Feet.	*Feet.*	*Pounds.*
1	97	19	1,000
2	92	21	900
3	93	22	850
4	89	21	850
5	78	20	750
6	78	20	750
7	76	18	650
8	74	18	500
9	75	18	500
10	71	22	500
11	76	29	550
12	80	23	700
13	82	25	750
14	85	20	750
15	86	26	750
16	88	24	800
17	90	27	850
18	92	23	800
19	95	27	950
20	95	25	900

The shot was fired by means of cordeau in two main lines without cross connections. As far as the writer could judge by ear and eye, the explosion was simultaneous. A mass of rock of approximately 46,080 tons was thrown down and was so completely broken that little secondary blasting was required. The total cost of explosive, including fuse, was about 3.7 cents per ton of rock obtained. Approximately 3 tons of rock were thrown down for each pound of explosive.

A description of one of the largest blasts ever made in a limestone quarry in the United States will be of interest. *A*, figure 17, is a plan showing the arrangement, burden, and spacing of the drill holes, and *B* is the profile showing the depths of the drill holes. There were 46 vertical holes of an average depth of 91 feet 11 inches, an average burden of 36 feet 5¾ inches, and an average spacing of 19 feet 7 inches. Eight horizontal "snake holes" were drilled at that part of the face where the burden was heaviest. The length of the face was 880 feet. The spacing, burden, depth, and amount of explosive for each drill hole are shown in Table 13.

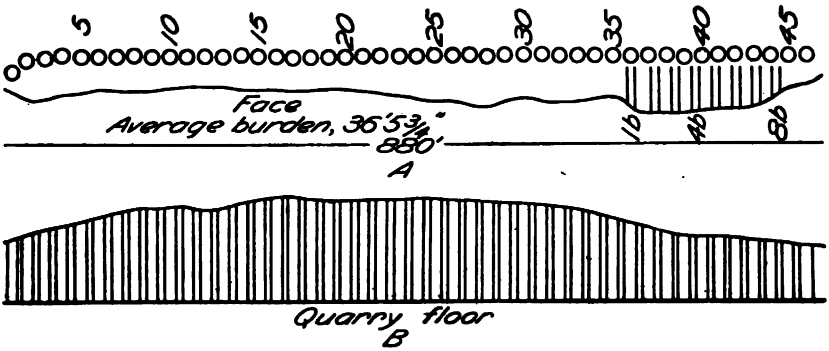


FIGURE 17.—Arrangement of drill holes for a large blast in a Tennessee quarry: *A*, Plan: *B*, profile.

TABLE 13.—*Data for a large blast in a limestone quarry.*

Hole No.	Spacing.	Burden.	Depth.	Explosives used.			Cost.
				60 per cent gelatin dynamite.	60 per cent straight nitroglycerin dynamite.	Low-freezing dynamite.	
	Ft. in.	*Ft. in.*	*Ft. in.*	*Pounds.*	*Pounds.*	*Pounds.*	
1		35 0	66 5	....	415	153	$70.90
	23 9						
2		38 0	70 1	....	414	208	76.89
	18 8						
3		38 0	75 0	....	315	263	70.10
	20 0						
4		37 0	77 6	....	463	208	83.26
	21 6						
5		35 0	81 5	101	411	155	83.74
	18 3						
6		37 0	86 3	166	515	102	99.84
	21 10						
7		35 0	91 4	313	316	99	92.73
	17 0						
8		35 0	91 11	359	360	199	115.53
	16 0						
9		34 0	96 9	314	311	45	98.00
	20 6						
10		33 0	98 7	259	323	151	80.66
	20 6						
11		32 0	97 9	354	276	101	93.10
	18 3						

TABLE 13.—*Data for a large blast in a limestone quarry*—Continued.

Hole No.	Spacing.	Burden.	Depth.	Explosives used.			Cost.
				60 per cent gelatin dynamite.	60 per cent straight nitroglycerin dynamite.	Low freezing dynamite.	
	Ft. in.	*Ft. in.*	*Ft. in.*	*Pounds.*	*Pounds.*	*Pounds.*	
12	21 3	31 0	100 1	408	279	49	$94.74
13	20 6	29 0	103 6	412	370	47	106.86
14	20 0	26 6	104 5	412	321	98	106.15
15	17 3	30 0	106 2	307	375	170	107.51
16	21 0	31 0	107 7	357	503	20	114.00
17	18 6	29 0	107 9	314	415	100	105.85
18	19 10	31 6	108 9	310	369	65	95.47
19	21 6	31 0	108 1	309	429	157	113.34
20	20 0	24 0	109 2	257	431	101	100.64
21	21 8	25 6	109 8	312	339	103	96.05
22	17 6	29 0	108 8	314	369	151	105.53
23	20 3	31 0	107 9	309	361	152	103.96
24	19 10	32 0	107 0	362	364	101	105.58
25	16 0	35 0	106 1	408	372	98	112.25
26	19 2	39 0	105 4	421	335	101	109.48
27	20 0	40 6	104 10	474	382	48	116.61
28	20 2	43 0	104 5	472	374	263	139.14
29	21 6	44 0	104 2	391	247	152	99.79
30	20 3	44 0	103 6	413	374	262	131.36
31	23 10	43 6	103 2	417	431	49	115.69
32	20 0	42 0	101 2	356	279	47	87.75
33	16 0	40 0	98 7	416	185	146	94.33
34	21 0	40 0	94 11	419	330	99	108.34
35	21 6	39 6	90 0	412	237	48	89.69
36	22 6	39 0	84 0	412	233	101	95.05
37	20 0	....	79 9	316	318	101	93.61
38	17 0	....	79 1	415	229	98	94.58
39	19 6	....	76 1	315	534	102	121.68
40	16 6	....	73 3	258	386	188	104.56
41	15 0	....	73 11	306	243	132	86.00
42	16 6	....	71 0	260	305	108	85.42
43	16 0	....	67 7	260	195	103	70.57
44	14 4	33 0	64 8	312	242	53	77.89
45	12 2	31 0	62 0	207	305	106	78.31
46	....	31 0	60 4	206	241	106	69.86
1*b*		....	32 0	259	194	210	82.16

TABLE 13.—*Data for a large blast in a limestone quarry*—Continued.

Hole No.	Spacing.	Burden.	Depth.	Explosives used.			Cost.
				60 per cent gelatin dynamite.	60 per cent straight nitroglycerin dynamite.	Low-freezing dynamite.	
	Ft. in.	*Ft. in.*	*Ft. in.*	*Pounds.*	*Pounds.*	*Pounds.*	
2B	14 8	....	35 6	311	336	160	$101.84
3B	13 0	....	35 0	419	238	453	135.61
4B	13 6	....	34 0	201	183	218	74.07
5B	15 0	....	29 0	206	198	220	76.91
6B	16 0	....	16 0	150	234	167	68.42
7B	14 4	....	17 0	200	306	109	77.86
8B	13 0	....	19 0	200	201	159	69.73
Total.	....	....	....	16,061	17,711	7,205	5,188.99

The 12 snake holes required 548 pounds of "60 per cent" gelatin dynamite. A total of 7,130 feet of cordeau was used, the cost of which was $304.95. The following summary indicates the high efficiency of the shot:

Results of shot.

Total weight of explosives, pounds	41,525
Total cost of explosives, including dynamite, fuse, and caps	$5,568.37
Weight of rock shot down, tons	242,096
Cost of explosive, cents per ton of rock moved:	
Primary shot	2.30
Secondary shots	1.37
Total cost of explosives	3.67
Quantity of rock moved per pound of explosive, tons	5.83

Plate IV shows the quarry face prior to the blast, and Plate V, the new face and the mass of shattered rock resulting from the blast. Plate VI illustrates the blast itself, photographed at the moment of detonation.

GENERAL CONSIDERATIONS IN BLASTING.

PROPER BALANCE BETWEEN EXPLOSIVES COST AND DRILLING COST.

A given quantity of rock may be broken with less explosive by drilling a large number of small holes closely spaced than with a smaller number of larger and more widely spaced holes. Therefore, within certain limits the cost of explosives evidently will decrease as the cost of drilling increases, and vice versa. If either factor is overemphasized, blasting efficiency is decreased. It is important, therefore, that the quarryman attain a proper balance between these two costs. The soundness of the rock, the rate of drilling, the ease or difficulty of shattering, the method of loading rock, and other considerations must all be taken into account. A ratio of drilling costs to explosives costs that might prove the cheapest in one quarry might not be suited to another quarry. The quarryman must determine by experiment the relative costs of drilling and explosive that will give the lowest aggregate cost of the two.

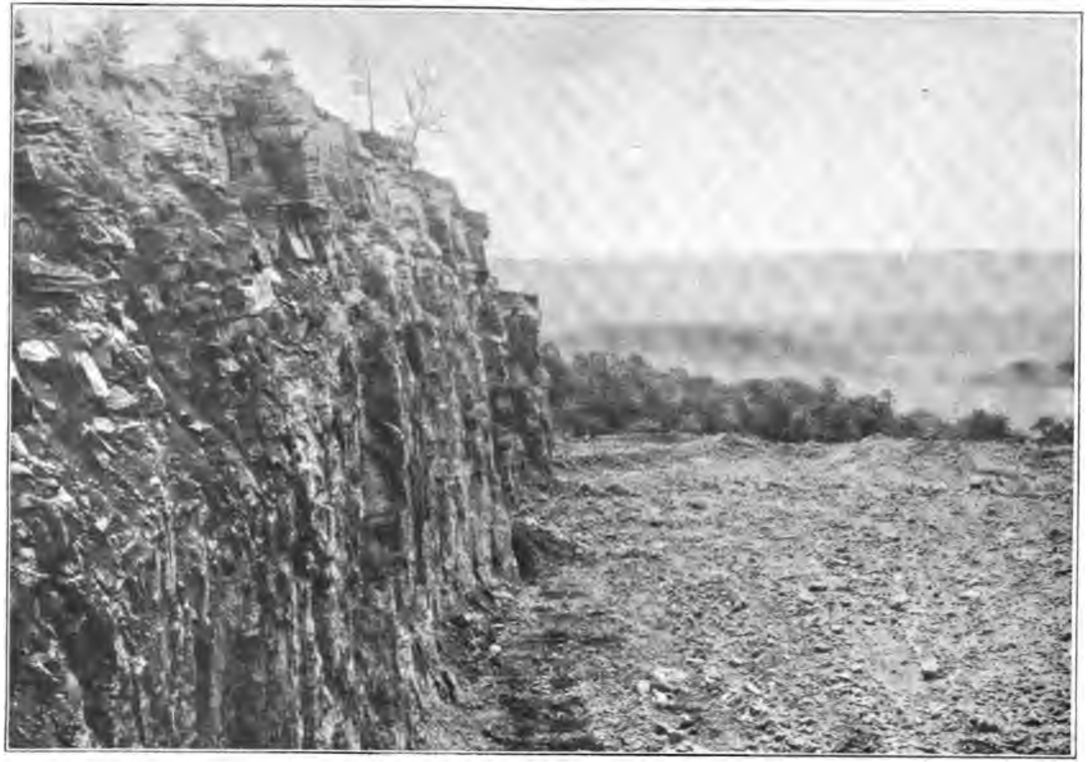
PROPER BALANCE BETWEEN BLASTING EFFICIENCY AND ROCK-LOADING EFFICIENCY.

All quarrymen know that the size of a charge must be so regulated that the rock will not be thrown too far. Aside from obstructing and damaging of tracks, a wide, thin mass of rock is more expensive to load with a steam shovel than a thicker mass which covers a smaller area. It may be assumed that loading can be done most efficiently when the face of the rock pile stands at its maximum angle of repose, that is, as steep as the fragments will lie without rolling down. This condition is most easily attained in comparatively low benches where "blanket" blasting may be employed. The buffer of shattered rock prevents fragments scattering and thus aids in maintaining a steep face. With a high face the proper adjustment of the charge, both as to spacing of drill holes and quantity of explosive used, is a more difficult matter.

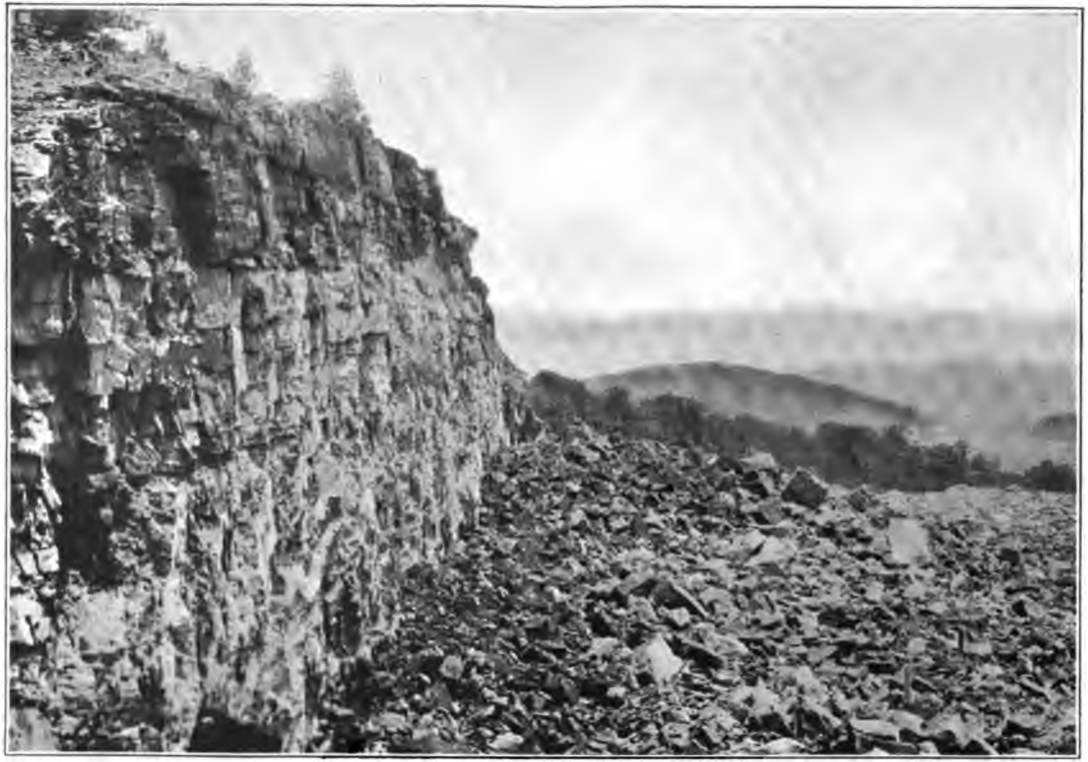
Moreover, there is an intimate relation between rate of loading and efficiency in blasting. If light charges are employed the explosives costs may be low, but the rock may be so imperfectly shattered that loading is repeatedly delayed while blocks ahead of the steam shovel are being broken by secondary blasting. Owing to such delays, the daily tonnage loaded with each shovel may be small.

PROPER BALANCE BETWEEN SIZE OF BLAST AND SIZE OF STEAM SHOVEL.

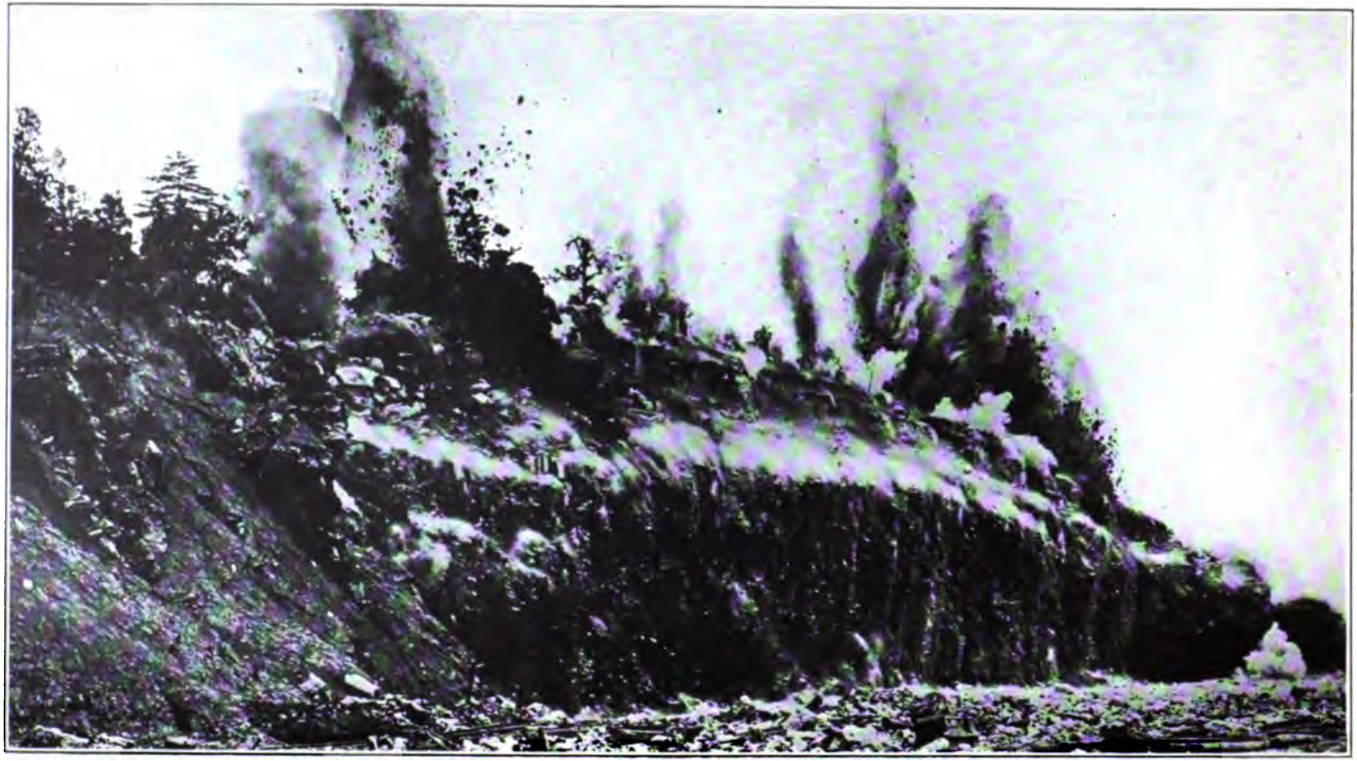
In some quarries where "blanket" blasting is employed on a low face, blasting may precede steam-shovel loading by a long period of time. Under such conditions the two operations are independent of each other insofar as size of blast and width of steam-shovel cut are concerned. In other quarries steam-shovel loading follows blasting closely. In blasting on a high face the rock brought down by one blast is all removed before another mass is shot down, but in "blanket" blasting on a low face, a definite width of shattered rock may be left for a buffer. In either case the position of the steam-shovel tracks for the final cut is guided by the position of the face. As pointed out on a subsequent page (p. 123), one of the necessary



FACE IN TENNESSEE QUARRY JUST BEFORE A LARGE BLAST.



APPEARANCE OF THE QUARRY AFTER THE BLAST.



BLAST PHOTOGRAPHED AT THE MOMENT OF DETONATION. NOTE THE "SNAKE-HOLE" SHOT, IN LOWER RIGHT-HAND CORNER.

conditions for efficient loading is that each cut shall be the maximum width the shovel is capable of handling. Therefore it is important that the mass to be removed is of such width that it may be taken in full cuts. Usually the mass is removed in two or three cuts. The actual number of cuts varies because it depends on the size of the shovel and the magnitude of the mass that may be shot down efficiently at one time, but it is important that the mass shall not require a narrow cut to complete its removal. For example, if the mass to be removed is 2½ cuts wide, the expense of moving the track and the number of moves for the shovel are the same for the half cut as for a full cut, consequently the cost of loading the half cut is excessive. Therefore the blasting foreman should so judge the magnitude of the charges that the mass of shattered rock resulting from the blast shall when lying at the maximum angle of repose be of such a width that it may be removed at one, two, or more full cuts. On the other hand if it is found that a blast of a certain size gives greater efficiency than those of larger or smaller size, then the size of the steam shovel should be such that it can load the entire mass in full cuts. The proper adjustment may therefore be made by varying either the size of the blast or the size of the steam shovel, the method of adjustment being governed by quarry conditions.

SECONDARY BLASTING.

DEFINITION OF SECONDARY BLASTING.

The term secondary blasting, as previously stated, is applied to the process of breaking with explosives the larger pieces resulting from the primary shots into sizes convenient for loading. The process is known locally as "blistering" or "bulldozing." Secondary shots are also occasionally termed "subsequent" shots.

CONDITIONS GOVERNING THE EXTENT OF SECONDARY BLASTING.

The amount of secondary blasting required depends on three main factors: (1) The nature of the rock, (2) the efficiency of the primary shot, and (3) the method of loading rock.

Rock that breaks easily and splits readily parallel with the bedding, like much of the cement rock of the Lehigh district, requires little secondary blasting, as the primary shots break the rock into small pieces. However, many limestone ledges break with difficulty, and many of the fragments resulting from the primary shot may be of large size. An excessive number of large blocks may also result from the presence of open joints which intersect the rock ledge and divide it into numerous independent masses. The masses in which the holes for the charges are drilled will probably be shattered sufficiently, but those having no drill hole in them will probably be only slightly shattered, because the seams act as buffers and tend to check the force of the explosion. As a result, such masses may be merely pushed out and may not be broken. As a rule, the drill holes for primary shots should be more closely spaced in seamy than in sound rock, for such an arrangement promotes shattering and decreases the amount of secondary blasting necessary.

Also, light charges tend to throw the rock down in large fragments, whereas heavier charges would probably break it up much better. Furthermore, a low-grade explosive breaks the rock less efficiently than a high-grade explosive. In a number of quarries, "60 per cent" dynamite is used for all primary shots, for it is claimed that such an explosive breaks the rock much better than "40 per cent" dynamite. In quarries where the rock is difficult to break into fragments of convenient size, the quarryman should aim to use explosives that have a high disruptive effect rather than those the effect of which is mainly propulsive.

Secondary blasting is influenced greatly by the method of loading the rock. Rock loaded by hand must be broken into sizes that one man can handle, or can easily be reduced by sledging to the desired size, whereas steam shovels can load fragments of large size. Thus the expense for secondary blasting is usually much higher where hand loading is employed than where steam shovels are used.

METHODS OF SECONDARY BLASTING.

Two methods are in common use in secondary blasting, known as the "mud-capping" or "adobe" method, and the "block holing" method. The mud-capping method is known locally by other names such as "dobeying," "blistering," or "bulldozing," though some of these terms are applied to any method of secondary blasting. The method consists in placing a stick of dynamite, with a fuse attached, on the surface of the rock to be broken, and covering the dynamite with a mass of clay which tends to confine the explosion and direct it toward the rock. In the block-holing method, holes several inches deep are drilled in the blocks with hammer drills, and a stick, or part of a stick, of dynamite is placed in each hole. The rock dust that results from drilling is commonly used for stemming. Comparatively low-grade explosive is employed, "30 per cent" dynamite being commonly used. Whatever method is employed, a number of blasts are prepared, the fuses lighted, and the blasts thus discharged in rapid succession. In one quarry in the Lehigh district, electric detonators are placed in the charges, connected by wires, and fired simultaneously with an electric current.

ADVANTAGES OF "BLOCK-HOLING" OVER "MUD-CAPPING" METHOD.

Mud-capping is usually regarded as an inefficient and expensive method of secondary blasting. It is well known that black blasting powder discharged on a rock surface has little or no effect on the rock, whereas dynamite discharged in the same manner may shatter the rock to a considerable extent. For this reason it is a common belief among quarrymen that the explosive force of black blasting powder is directed upward, whereas that of dynamite is directed downward. It should be clearly understood, however, that all explosives exert equal force in all directions. The different results obtained from black blasting powder and dynamite are due to the different rates of combustion of the two explosives. Gases are evolved at a comparatively low rate of speed from black blasting powder and, consequently, they can readily push aside the overlying air and escape. When dynamite is discharged, however, the gases are evolved so rapidly that they can not readily escape and, consequently, exert a violent shattering force upon any body with which the dynamite is in contact. As this force is the same in all directions, it may be readily understood that when a stick of dynamite is discharged on the surface of a rock, a large proportion of the energy of the explosion is wasted and does no useful work. On the other hand, if the dynamite is discharged in a drill hole, a much greater proportion of the energy is expended on the rock.

As a result of tests made by the Bureau of Mines,[a] it was found that the block-holing method is at least 10 times as effective in breaking limestone as the adobe method for a given quantity of explosive. Similar conclusions have been reached by many quarrymen as a result of practical experience. As a consequence, the block-holing method is commonly employed for a major part of secondary blasting, the adobe method being used only under special conditions. In some places, however, the latter method is employed extensively and with low efficiency.

METHOD OF BLOCK-HOLE BLASTING.

In drilling for secondary shots, lines of hose are usually employed to conduct compressed air to the hammer drills from the main air lines on the quarry bank. Other methods have also been noted. In one quarry a motor-driven air compressor is mounted in a box car and can thus be readily shifted to the place where drilling is required, making long air lines unnecessary. In another quarry the steam shovel is equipped with a compressor of the Westinghouse air-brake type. Where secondary blasting is carried on within a short distance of the steam shovel, this is a very convenient method.

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[a] See Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1912, p. 15; Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 48, Bureau of Mines, 1913, pp. 37–39.

If many large blocks are visible in the mass of rock thrown down by the primary shot, it is customary in some quarries to drill and blast these blocks ahead of the steam shovel. Where the mass is broken up more thoroughly by the primary shot such preliminary blasting is usually omitted. Even where the primary blasting is highly efficient blocks of large size are frequently encountered in loading with a steam shovel. Some of the blocks may be too large to be lifted with the dipper, or blocks that the steam shovel could handle may be too large for the crusher. In either event, it is customary to push such blocks to one side. As the steam shovel is advanced these large masses are left between the steam-shovel track and the face. They are later drilled and blasted and the resulting fragments are loaded with the next cut.

Where hand loading is employed the larger masses exposed at the surface are first blasted and other large masses are blasted subsequently as they are encountered in loading.

CIRCUMSTANCES THAT JUSTIFY THE MUD-CAPPING METHOD.

Occasionally in steam-shovel loading masses of rock too large to be moved with the dipper may effectively block further progress. For such blocks "mud-capping" is probably preferable to "block holing" to save time. The drilling might require only a few minutes, but during this time the steam-shovel gang and the train crew would be idle. The use of a few more pounds of explosive is cheaper than to keep 10 or 12 men, a locomotive, and a steam shovel idle for even as short a space as 20 minutes or one-half hour.

Another circumstance which may justify mud-capping is the obstruction of tracks with blocks of stone. As a result of a blast or a rock slide, large masses of stone may roll onto the tracks, preventing the passage of cars or locomotives. Stalling of loaded or empty cars may not only hinder the train crew, but also keep the steam-shovel gang or the hand loaders idle for lack of cars. As in the preceding case, time is of more value that a few pounds of dynamite. Therefore, mud-capping, which requires only a few minutes, is more economical than block-holing.

Where the large fragments are thin and slablike in form it may be difficult to obtain drill holes of effective depth, and mud-capping may give better results than the block-holing method.

The amount of blasting under the special conditions that may justify mud-capping, as outlined in the preceding paragraphs, probably represents a very small proportion of the total secondary blasting required in any quarry. Except when these conditions arise, all secondary blasting should be by block-holing.

EXAMPLES OF EXCESSIVE SECONDARY BLASTING COSTS.

A number of instances have been noted where the cost of secondary blasting was excessive. It may be instructive to quarrymen to review the probable causes of the high costs, in order that inefficient methods may be recognized and avoided.

In one limestone quarry observed, more explosive is used for secondary than for primary blasting. Two causes may be assigned. First, the rock is seamy, and hence many large, solid masses are thrown down. Where such conditions prevail, the quarryman may find some useful suggestions on pages 50–51, where blasting in unsound rock is discussed. In the second place, the high cost is partly due to hand loading, for the rock must be broken into small pieces to be loaded by this method.

In a limestone quarry in the Middle West, excessive secondary blasting is required because the rock is very hard and is thrown down in large solid fragments by the primary shots. Tripod-drill holes are placed 5 feet apart and with a 6-foot burden on 32-foot benches. The explosive employed is 40 per cent "straight" nitroglycerin dynamite. Larger and more widely spaced holes made with a churn drill or wagon drill would probably give better results, especially if "60 per cent" dynamite were employed.

In one southern limestone quarry as much explosive is used for secondary as for primary blasting. Apparently, this is due chiefly to the fact that the rock is loaded by hand and must be broken into pieces much smaller than is necessary for steam-shovel loading.

In several quarries visited by the writer, the high cost of secondary blasting is due chiefly to the employment of the wasteful and inefficient mud-capping method. The introduction of the block-holing method would greatly reduce the amount of explosive required.

EXAMPLES ILLUSTRATING BLASTING EFFICIENCY.

The following selected examples are fairly representative of the degree of efficiency attained in American quarries where rock is obtained for the manufacture of Portland cement. Conditions vary so widely that the results obtained in different places can not be intelligently compared without understanding the controlling conditions. Consequently, the conditions in each quarry are briefly outlined. The results are expressed as the number of tons of rock obtained per pound of explosive, and not in terms of costs. When given in this form, the result is independent of fluctuations in price of explosive.

EXAMPLE 1.

Conditions.—Limestone is thin-bedded, with a few open joints; churn-drill holes are 5 inches in diameter and 90 feet deep, with 17-foot spacing and 26-foot burden, and are arranged in a single row, 12 to 20 holes being fired at one time; continuous charge of "40 per cent" dynamite is used, with three electric detonators in each hole; secondary shots are chiefly by block-holing method; steam-shovel loading.

Result.—Approximately 4 to 5 tons of rock is obtained per pound of explosive, including secondary shots.

EXAMPLE 2.

Conditions.—Limestone dips 35° to 40°; has some open bedding planes and a few joints; quarry face is perpendicular to strike; churn-drill holes are 60 to 80 feet deep, with 16-foot spacing and 20-foot burden, 12 to 20 holes being fired at one time in single row; charge is continuous, "60 per cent" dynamite being used in bottom of hole and "40 per cent" dynamite for main part, and is fired with detonating fuse; rock is loaded by hand.

Result.—Approximately 3½ to 4 tons of rock is obtained per pound of explosive, not including secondary blasting.

EXAMPLE 3.

Conditions.—Cement rock is thin-bedded in upper 20 feet, with solid ledge beneath; churn-drill holes are 145 feet deep, with 18-foot spacing and 20-foot burden, 10 to 22 holes being fired at one time in single row; continuous charge, mainly "40 per cent" dynamite, "30 per cent" dynamite used near top, four-electric detonators used in each hole; steam-shovel loading.

Result.—About 3¾ tons of rock is obtained per pound of explosive, including secondary shots.

EXAMPLE 4.

Conditions.—Limestone lies flat, is thin-bedded near top, but otherwise sound; 20-foot face; 8 to 20 5-inch churn-drill holes arranged in two rows and staggered are fired at one time; continuous charge of "40 per cent" ammonia dynamite, fired with electric detonators; steam-shovel loading.

Result.—Average for several years is 3.9 tons of rock per pound of explosive, including secondary shots.

EXAMPLE 5.

Conditions.—Limestone lies flat, is thin-bedded but fairly sound; 60 to 70 churn-drill holes 19 feet deep are shot at one time, the holes being in two. three, four, or six rows, and staggered; "blanket" blasting; continuous charge of "60 per cent" nitrostarch blasting powder, fired with two No. 8 detonators in each hole; steam-shovel loading.

Result.—About 4 tons of rock is obtain per pound of explosive, including secondary shots.

EXAMPLE 6.

Conditions.—Limestone dips at low angle, is thin-bedded, with few open and many closed seams; 25-foot face; about 20 5½-inch holes, with 10-foot spacing, arranged in two rows and staggered are fired at one time, supplemented by tripod drill holes in hard bed; broken charge of 40 per cent "straight" nitroglycerin dynamite is used, with intermediate stemming in soft bed, two electric detonators are placed in bottom charge, and one in top charge; steam-shovel loading.

Result.—About 3 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLE 7.

Conditions.—Limestone lies flat, is thin-bedded near surface, thick-bedded below; 30 to 35 foot face; 20 to 30 churn-drill holes, with 12-foot spacing and 12-foot burden, arranged in two rows and staggered are fired at one time; continuous charge of "60 per cent" gelatin dynamite, with electric detonators, is used; secondary blasting is mostly by "mud-capping" method.

Result.—About 2.7 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLE 8.

Conditions.—Limestone lies flat, is thick-bedded, with few seams; 22-foot face; an overburden of clay 2 to 4 feet thick is shot down with rock and included in tonnage; about 50 4-inch churn-drill holes, with 7 to 10 foot spacing and 12-foot burden, arranged in two rows and staggered are fired at one time with electric detonators, 60 per cent "straight" nitroglycerin dynamite used; steam-shovel loading.

Result.—About 2¼ tons of rock is obtained per pound of explosive, primary blasting only.

EXAMPLE 9.

Conditions.—Limestone lies flat, is thick-bedded; 30 to 50 foot face; 20 to 30 churn-drill holes with 18-foot spacing and 18-foot burden are shot at one time in one row, with occasional holes closer to face and staggered; 60 per cent "straight" nitroglycerin dynamite and electric detonators are used for primary shots; secondary blasting is nearly all by block-holing method, using hammer drill and "40 per cent" dynamite; steam-shovel loading.

Result.—About 2.6 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLE 10.

Conditions.—Limestone lies nearly flat, is thick-bedded, has some open bedding seams and some open joints; 70 to 80 foot face; 20 to 40 churn-drill holes arranged in two rows and staggered are fired at one time; first row has 18-foot spacing and 15-foot burden, second row has 18-foot spacing and 14-foot burden; for primary shots broken charge of 60 per cent "straight" nitroglycerin dynamite with intermediate stemming, fired with detonating fuse; secondary shots are mostly by block-holing method, "40 per cent" dynamite being used; steam-shovel loading.

Result.—About 4 tons of rock is obtained per pound of explosive used for primary blasting only, and 3.6 tons per pound of explosive including that used for secondary blasting.

EXAMPLE 11.

Conditions.—Limestone beds dip about 30°, have a few open seams and clay pockets; 80-foot face; 18 to 40 churn-drill holes are drilled in single row, with 20-foot spacing and 20-foot burden, holes extend 3 to 5 feet below quarry floor; a little "80 per cent" dynamite is used in bottoms of holes, main part of charge is "60 per cent" dynamite and "40 per cent" dynamite, electric detonators are used; rock is loaded by hand.

Result.—About 5 tons of rock is obtained per pound of explosive, primary shots only, and 2½ tons, including secondary blasting.

Example 12.

Conditions.—Limestone beds lie nearly flat, upper 20 to 30 feet are jointed. but rock beneath is sound; 40 to 60 churn-drill holes with an average depth of 95 feet, arranged in one row, or two rows staggered, spacing 16 to 22 feet, burden 36 feet, are fired at one time; charge is mostly "60 per cent" low-freezing dynamite, about one-fifth of the charge being of lower grade, and is fired with detonating fuse; steam-shovel loading.

Result.—About 5.8 tons of rock is obtained per pound of explosive, primary blasting only.

Example 13.

Conditions.—Limestone is thick bedded and dips 10° to 15°; 24 churn-drill holes 5⅝ inches in diameter and 56 to 121 feet deep, with a spacing of 16 to 17 feet and a burden of 22 feet; also 34 "snake holes" 16 feet long are drilled; churn-drill holes contain broken charges with 12 feet of intermediate stemming, "60 per cent" gelatin dynamite and "30 per cent" dynamite are used in about equal quantities, the charge being fired with detonating fuse.

Result.—About 4.2 tons of rock is obtained per pound of explosive, primary blasting only, rock well broken.

Example 14.

Conditions.—Limestone beds lie flat, have many clay seams; 50-foot face; 50 per cent "straight" nitroglycerin used in 5⅝-inch churn-drill holes, burden and spacing variable on account of seams; steam-shovel loading.

Result.—About 3.8 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLES OF QUARRY COSTS.

The following additional examples of actual quarry costs may be of interest:

Example 1.

Conditions.—Bench quarry, no stripping, limestone dipping at a high angle, numerous open bedding planes; tripod drills employed, making holes 2½ inches in diameter and 16 feet deep; spacing 6 to 10 feet; "40 per cent" dynamite used; secondary blasting is by both "block-holing" and "mud-capping" methods, more explosive being used in secondary than in primary blasting; rock is hand-loaded; cars are hauled by mules.

Result.—Operating cost of delivering rock to crusher, 18 to 19 cents per ton.

The use of more explosive for secondary than for primary blasting is due to the fact that the useful work done by the explosive in the primary shots is lessened by the presence of open seams, and also to the necessity of breaking the rock into small pieces suitable for hand loading.

Example 2.

Conditions.—Crystalline limestone, overburden 6 to 15 feet of clay or shale, many large clay pockets, rock variable in composition, churn-drill holes are 15 to 20 feet deep and 6 inches in diameter, explosive used is "40 per cent" ammonia dynamite, electric detonators; secondary shots are mostly by block-holing method; rock is hand-loaded on four levels, locomotive transportation on heavy grades.

Result.—Operating cost of delivering rock to crusher, stripping included, is 32 cents per ton.

EXAMPLE 3.[a]

Conditions.—Quarry is situated in the Middle West; rock is an argillaceous limestone, fairly uniform in quality, with occasional seams and pockets; face is 40 to 60 feet high, and is shot down as a single bench; churn drill employed, drilling holes 6 inches in diameter and averaging 55 feet in depth, with a spacing of 10 feet and a burden of 8 feet; about 14 holes are shot at one time, "40 per cent" gelatin dynamite is used, about 150 pounds in each hole; rock is loaded with steam shovels.

Result.—The total operating cost of drilling, including repairs, is 3.15 cents per ton of rock. Assuming a price of 12 cents a pound for the explosive, the cost of blasting per ton of rock is 5.76 cents. The total cost of drilling and blasting is, therefore, 8.91 cents per ton of rock produced.

EXAMPLE 4.

Conditions.—Rock is hard limestone, with 3 to 8 feet of soil overburden; churn drills are used, drilling holes 5⅝ inches diameter and averaging 70 feet deep, main charges are 40 per cent "straight" nitroglycerin dynamite, with a little "60 per cent" dynamite in bottoms of holes; secondary blasting is mostly by "mud-capping" method; rock is loaded with three tractor shovels having dippers of ¾-yard capacity.

Result.—Cost of explosives (dynamite at 20 cents per pound) is 7 cents per ton of rock obtained; total operating cost of delivering rock to crusher is about 30 cents per ton.

EXAMPLE 5.

Conditions.—Limestone, flat-lying, thin-bedded, 25-foot face; blasts in churn-drill holes, in two rows and staggered, 8 feet apart, "40 per cent" ammonia dynamite used, buffer method of blasting is employed; secondary blasting is by block-holing method; rock is loaded by steam shovel; locomotive haulage; average output about 1,500 tons per day.

Result.—Total operating cost of delivery to crusher, stripping not included, is 12 to 13 cents per ton of rock obtained.

BLASTING RECORDS.

IMPORTANCE OF KEEPING RECORDS.

The preceding pages have shown that blasting is a complicated problem involving many interdependent factors. The progressive quarry operator constantly endeavors to reduce the cost of production. As a result, many changes in methods of blasting may be made, and the only way in which the operator can judge the relative efficiency of the various methods tried is to keep a careful cost record of each. Therefore the maintenance of such records is an essential factor in successful blasting.

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[a] See McDaniel, A. B., Methods employed in drilling and blasting in the cement quarry: Concrete Cement Age, June, 1914, p. 70.

All records should be reduced to some convenient basis for comparison. Drilling may be expressed as cost per foot of hole drilled, and explosives as the amount of rock moved per pound of explosive used, or as the cost per ton of rock obtained. Usually all costs are reduced to a rock tonnage basis.

EXAMPLES OF BLASTING RECORDS.

The following tables are actual blasting records from a Pennsylvania quarry. Three records are given in order that the relative efficiency of the different methods may be compared.

TABLE 14.—*Examples of blasting records kept at a Pennsylvania quarry.*

BLAST 11.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Primary blast.				
Labor:				
Drilling 624 feet, at 44 cents........ $274.56				
815 feet, at 42 cents........ 342.30				
	$616.86	1.62		
Cleaning holes	5.85	.02		
Handling powder and loading holes	30.58	.08	$653.29	1.72
Supplies:				
Dynamite, low-freezing—				
7,606 pounds, 30 per cent, at 9 cents per pound	684.54	1.80		
200 pounds, 40 per cent, at 11 cents per pound	22.00	.06		
1,350 pounds, 50 per cent, at 11 cents per pound	148.50	.39		
500 pounds, 60 per cent, at 12½ cents per pound	61.25	.16	916.29	2.41
Detonators—				
12, 40-foot	1.55			
10, 80-foot	2.38	.01		
11, 100-foot	3.20	.01		
13, 120-foot	4.50	.01	11.63	.03
Total, primary blast	....	....	1,581.21	4.16
Secondary blasting.				
Labor:				
Air drilling	14.17	.04		
Blasting	84.30	.22	98.47	.26
Supplies:				
Dynamite, 700 pounds, 40 per cent	77.00	.20	77.00	.20
Fuses, 1,100 feet	3.52	.01		
Detonators, 125 pieces	.88	....	4.40	.01
Total, secondary blasting	....	....	179.87	.47
Grand total	....	....	1,761.08	4.63

GENERAL DATA.

Holes drilled, 12.
Drilling hours, 419.
Feet drilled, 1,439.
Feet drilled per hour, 3.43.
Rock blasted, 37,994 gross tons.
Tons per pound of dynamite, primary blasting, 3.93.
Blast made July 1, 1915.
Drilling done by contract, 100 per cent.

TABLE 14.—*Examples of blasting records kept at a Pennsylvania quarry*—Con.

BLAST 12.

Labor and material.		Itemized cost.		Total cost.	
		Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Labor:					
Contract drilling—					
337 feet, at 42 cents	$141.54				
1,608 feet, at 40 cents	643.20				
Drilling by company, 512 feet	25.00				
		$809.74	1.13		
Cleaning holes		3.80	.01		
Handling powder and loading holes		32.33	.04	$845.87	1.18
Supplies:					
Nitrostarch blasting powder—					
5,050 pounds, 35 per cent, at 9⅗ cents per pound		484.80	.68		
15,900 pounds, 40 per cent, at 10 cents per pound		1,590.00	2.21		
1,650 pounds, 50 per cent, at 11 cents per pound		181.50	.25	2,256.30	3.14
Detonators—					
20, 50-foot		3.75	....		
20, 80-foot		5.63	.01		
40, 100-foot		13.80	.02		
Connecting wire and tape		1.15	....	24.33	.03
Total, primary blast		....	....	3,126.50	4.35
Secondary blasting.					
Labor:					
Air drilling		31.40	.04		
Blasting		144.50	.20	174.90	.24
Supplies:					
Dynamite, 900 pounds, 40 per cent		90.00	.13	90.00	.13
Fuses, 1,700 feet		5.44	.01		
Detonators, 200 pieces		1.40	....	6.84	.01
Total, secondary blasting		....	....	271,74	.38
Grand total		....	....	3,398.24	4.73

GENERAL DATA.

Holes drilled, 20.
Feet drilled by contract, 1,945 feet in 477 hours, 4.08 feet per hour.
Feet drilled by company, 512 feet in 100 hours, 5.12 feet per hour.
Rock blasted, 71,872 gross tons.
Tons per pound of explosive, primary blast, 3.18.
Blast made August 23, 1915.
Drilling done by contract, 79 per cent; by company, 21 per cent.

BLAST 14.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Primary blast.				
Labor:				
Drilling by company, 1,850 feet	$137.91	0.31		
Cleaning holes	5.40	.01		
Handling powder and loading holes	24.99	.06	$168.30	0.38
Supplies:				
Nitrostarch powder—				
4,500 pounds, 50 per cent, at 11 cents per pound	495.00	1.13		
10,000 pounds, 30 per cent, at 9⅕ cents per pound	920.00	2.10	1,415.00	3.23
Cordeau, 5 spools	114.66	.26		
Detonators, 1 piece	.12	....	114.78	.26
Total, primary blast	....	....	1,698.08	3.87

TABLE 14.—*Examples of blasting records kept at a Pennsylvania quarry*—Con.

BLAST 14—Continued.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Secondary blasting.				
Labor:				
Air drilling	$24.00	0.05	$151.03	0.34
Blasting	127.03	.29		
Supplies:				
Dynamite, 800 pounds, 40 per cent, at 10.08 cents per pound	80.69	.18	80.69	.18
Fuse, 2,300 feet	8.35	.02	10.89	.03
Detonators, 300 pieces	2.54	.01		
Total, secondary blast			242.61	.55
Grand total			1,940.69	4.42

GENERAL DATA.

Holes drilled, 15.
Feet drilled, 1,850.
Drilling time, 284 hours.
Feet per hour, 6.51.
Rock blasted, 43,945 gross tons.
Tons per pound of explosive, primary blast, 3.03.
Blast made February 1, 1916.
Drilling done by company, 100 per cent.

A noteworthy feature of the records is the information they convey regarding drilling costs. As noted in the general data for blast 11, all the drilling for that blast was done by contract. The cost of drilling was 1.62 cents per gross ton of rock obtained. For blast 12, when 79 per cent of the drilling was done by contract and 21 per cent by the quarry company, the cost was reduced to 1.13 cents per ton. For blast 14, in which all drilling was done by the company, the cost per ton was only 0.31 cents.

Similar comparisons may be made of the number of feet drilled per hour by each method, the effect on costs of the use of cordeau in blast 14 as compared with the use of electric detonators in blasts 11 and 12, and of other factors.

The chief purpose of such records is to provide the operator with information for intelligently comparing methods and determining which give the best results.

A complete blasting record should include not only a statement of labor and supplies for drilling and loading, but also a description of the blast, indicating number, depth, and arrangement of drill holes, and amount of explosive in each. The following record of a blast in a limestone quarry is given as an example. The form of the report may be modified to suit the quarry conditions. With such records before him, the quarry operator may deduce valuable information regarding the type of explosive and size of charge that does the best work, and the most efficient burden and spacing of drill holes.

TABLE 15.—*Form of daily blasting report adopted by a Michigan limestone quarry company.*

Fired 2.40 p. m., Oct. 13, 1916.

Hole No.	Burden.	Depth.	Dynamite used.	Per cent.	Kind.	Depth of stemming.	Fuses.		Dynamite, top load.	Per cent.	Kind.	Fuses.		Remarks.
							Number.	Length.				Number.	Length.	
	Feet.	*Feet.*	*Pounds.*			*Feet.*			*Pounds.*					
1	22	55	200	40	Ex.	41	1 1	56 50	33	20	FFF.	1	30	Total dynamite used, 2,836 pounds.
2	23	50	200	40	Ex.	38	1 1	56 50	28	20	FFF.	1	30	
3	23	51	200	40	Ex.	38	1 1	56 50	28	20	FFF.	1	30	
4	23	55	200	40	Ex.	40	2—	50	28	20	FFF.	1	30	Total short tons rock moved, 21,182.4.
5	23	55	200	40	Ex.	39	1 1	56 50	32	20	FFF.	1	30	
6	23	55	200	40	Ex.	35	1 1	56 50	32	20	FFF.	1	30	
7	23	55	225	40	Ex.	35	1 1	56 50	28	20	FFF.	1	30	7.4 tons per pound of dynamite.
8	23	55	225	40	Ex.	34	1 1	56 50	33	20	FFF.	1	30	
9	23	55	215	40	Ex.	37	2—	50	28	20	FFF.	1	30	
10	23	55	240	40	Ex.	38	2—	46	33	20	FFF.	1	30	
11	23	54	230	40	Ex.	37	2—	46	33	20	FFF.	1	30	
12	20	54	165	40	Ex.	39	1 1	50 46	....	....	....	....	....	
Total or average	....	....	2,500	....	....	....	7 12 5—	56 50 46	336	....	....	12	30	

....

Blaster.

SAFETY IN BLASTING.

Manufacturers of explosives are familiar with the dangers involved in handling or storing them, consequently strict observance of the printed directions that accompany boxes of explosives is essential for safety.

Various safety precautions particularly applicable to limestone and cement-rock quarrying have been mentioned incidentally in the preceding pages. All men in charge of the handling or care of explosives at quarries should be familiar with the various publications[a] of the Bureau of Mines relating to safety in blasting. Constant vigilance is the price of safety in blasting, and all quarry operators should insist on shot firers being thoroughly familiar with all important safety rules and adhering strictly to them.

MINING ROCK FOR CEMENT MANUFACTURE.

EXTENT OF MINING.

Limestone and shale are usually worked by open-pit methods. However, in several places conditions are such as to justify underground working, which is herein termed mining to distinguish it from open-pit quarrying. Mining of raw materials for cement manufacture has been observed by the writer near Ironton, Ohio; Superior, Ohio; Independence, Mo.; Hannibal, Mo.; Wampum, Pa.; Manheim, W. Va.; Richard City, Tenn.; and Oglesby, Ill.

ADVANTAGES OF MINING ROCK.

Where the rock sought has little or no overburden open-pit quarrying is unquestionably cheaper than any mining method. As the depth of overburden increases, however, the cost of stripping increases proportionally, until a point is reached where mining would be cheaper if conditions are such that it can be substituted for the open-pit method. The greatest advantage of mining over open-pit quarrying is that the cost of stripping is saved. This saving is offset by the increase in cost of getting out the rock, and the quarryman must carefully balance these two factors in order to determine which method will be the cheaper.

A second advantage of mining is that the workmen are protected from inclement weather, and no time need be lost through adverse weather conditions, as in open-pit quarrying. As rain or snow can not enter the mine, the rock is dry except for underground water and condensation from the mine air.

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[a] See Munroe, C. E., and Hall, Clarence, A primer on explosives for metal mines and quarrymen: Bull. 80, Bureau of Mines, 1915, 125 pp.; Hall, Clarence, and Howell, S. P., Magazines and thaw houses for explosives: Tech. Paper 18, Bureau of Mines, 1912, 34 pp.; Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, 48 pp.

Another advantage is that beds deep enough to be worked by mining usually have no pockets or seams of clay, therefore soil does not get mixed with the rock.

DISADVANTAGES OF MINING ROCK.

One disadvantage in mining rock is the high operating cost per ton of rock obtained. Drilling and blasting costs are higher than in open-cut work, and are becoming relatively higher each year, because the cheaper method of deep-hole blasting, now in general use in quarries, can not be applied in mines. In mines the rock must usually be loaded by hand, although in rare instances power shovels may be employed. Thus in most mines loading is more expensive than in open-pit quarries. The high operating cost is further increased by the expense for lighting and ventilation. Also, the men are subject to dangers from falls of rock from the roof. The extent of this danger depends on the nature of the roof and the care that it receives.

CONDITIONS FAVORING ROCK MINING.

In open-pit quarrying only a limited thickness of overburden can be economically removed. Where the overburden is so thick that open-pit methods would be unprofitable, the deposit must be worked by some method of mining or else remain unused. In order that mining may be safe and practicable certain favorable conditions must be present. The most important consideration is the character of the material that will form the roof. In general, the expense of timbering in limestone or shale would be so high that mining could not be conducted profitably. Consequently the roof should be strong enough to require no support other than pillars, and the rock that overlies the serviceable stone should be strong and sound.

Shafts have been observed at some rock mines, but entries are more common, for they permit cheaper removal of the rock. Beds that lie approximately flat are the most desirable for mining, as most rock breaks more smoothly parallel with the bedding, and a smooth, nearly level floor simplifies transportation problems.

METHODS OF ROCK MINING.

DEFINITION OF TERMS EMPLOYED.

In describing mining operations it is necessary to use terms that are not employed in connection with open-pit operations. Some of the more common technical terms used in mining rock for cement manufacture are defined in the following paragraphs.

Mine.—The term mine as used herein is applied to underground workings. It is used in contrast with the "open-pit" quarry.

Roof.—The roof is the undisturbed rock that overlies the workings.

Entry.—An entry is an approximately horizontal passage through which access is gained to the mine and through which the rock is taken.

Shaft.—The term shaft is employed where the opening to the mine is vertical or steeply inclined.

Drift.—The term "drift" as used herein refers to a chamber or room.

Breast.—The breast is the working face at the end of a drift.

Bench.—The breast is usually worked in successive steps, each of which is termed a bench.

Pioneer bench.—The pioneer bench is the first bench that is blasted. It is usually the bench nearest the roof of the drift.

Crosscut.—A crosscut is a passage connecting contiguous drifts.

MINING SYSTEMS.

Limestone and shale mines are usually worked on a plan somewhat resembling the "room-and-pillar" system, and consist of a main entry and a series of parallel drifts, or chambers, connected by crosscuts. Supplementary air passages may be necessary to insure adequate ventilation.

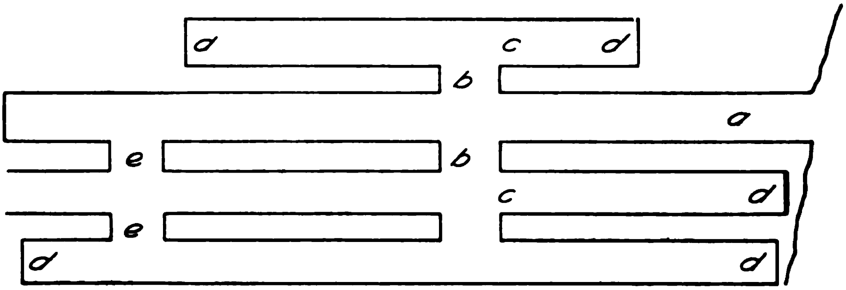

FIGURE 18.—Plan of workings in West Virginia limestone mine: *a*, Main entry; *b*, drift neck; *c*, drift; *d*, breast; *e*, crosscuts.

In some mines crosscuts are driven at such intervals that pillars about 20 feet square are left to support the roof. The drifts vary from 30 to 60 feet in width, depending on the strength of the roof. Thirty-five feet is the maximum height of drifts observed by the writer.

In a West Virginia mine the workings at the time of the writer's visit consisted of 10 parallel drifts, each 45 feet wide and 26 feet high, the pillars being about 30 feet thick. The rock mined is overlain with sound limestone 20 feet thick, which makes a secure roof.

In opening a new drift, a neck is turned off the entry and the drift is worked both to right and left, parallel with the entry, as shown in figure 18. This method provides for a large number of working places. At the time of visiting this quarry, mining was carried on at

16 breasts. Crosscuts for access from one drift to another and for ventilation are driven every 100 to 150 feet. The position of the crosscuts is governed to some extent by the position of open vertical cross joints; for, wherever possible, a joint plane is utilized to form one wall of the crosscut, as such a joint facilitates blasting.

At this mine the beds dip 17 feet in each 100 feet. The direction of the drifts with reference to the dip is important. The relative direction of dip, strike, and drift are illustrated in figure 19. The drift, although nearly parallel with the strike, runs up the dip at a low angle from the entry. This arrangement has two distinct advantages—gravity drainage and easy haulage for loaded cars. The lateral dip in the drifts has certain disadvantages, as the work of drilling holes and loading the rock is somewhat more difficult than on a level floor. Moreover, all tracks must be graded to a level bed, and this requires considerable labor. A vertical cross section of the drift workings is shown in figure 20.

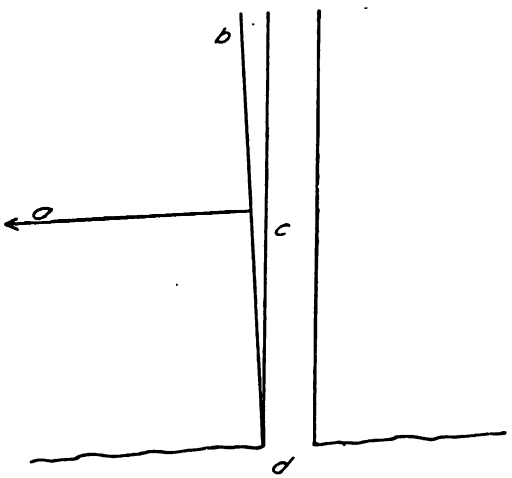

FIGURE 19.—Direction of drift with reference to dip and strike, in a West Virginia limestone mine: *a*, Dip; *b*, strike; *c*, drift; *d*, entry.

In an Illinois limestone mine the pillars at one side of the main entry are opposite the rooms, or chambers, at the other side. It is claimed that when the pillars are staggered in this way the roof holds much better.

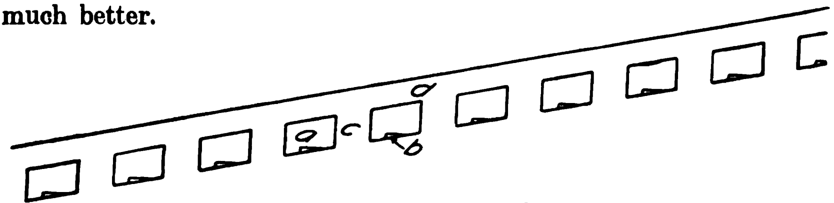

FIGURE 20.—Vertical cross section of drifts: *a*, Drift; *b*, track; *c*, pillar; *d*, roof.

The dimensions of the main entry, crosscuts, air course, and chambers in a Tennessee shale mine are given in a previous paragraph. On account of the weakness of the shale the pillars between the chambers are 50 to 75 feet wide.

SHAFT MINES.

Limestone mines with shafts are rare. The only one of this type observed by the writer is in southern Ohio, where a bed of very purlimestone is worked through a shaft over 500 feet deep. The rock is mined by the room-and-pillar system and is hoisted to the surface. A current of air downcast by fans through a second shaft provides the necessary ventilation.

DRIFT MINES.

The more common method of opening a limestone mine is by horizontal entry. Some mines have a single entry from which various drifts or chambers are turned off; in others access may be had through a series of entries. Transportation of rock is easier through a level or slightly sloping drift than through a shaft. Also, where the underground workings are reached by approximately level entries, drainage is usually simplified and is commonly by gravity.

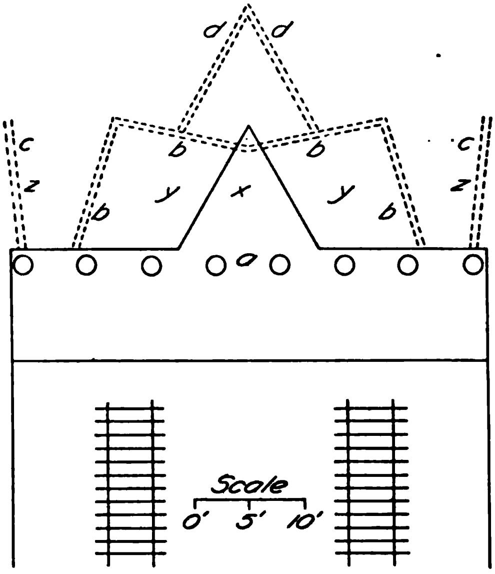

Figure 21.—Method of working breast in a West Virginia limestone mine: *x*, V or core; *y*, shoulders; *z*, sides; *a*, blast holes for 12-foot bench; *b*, blast holes for shoulders; *c*, blast holes for removing sides; *d*. blast holes for removing core.

METHODS OF DRIVING DRIFTS.

The usual method of working the breast in drifts is to blast out a pioneer bench, the top of which is at or near the roof level, and then shoot down the rest of the breast in one or more following benches. Details of methods of driving drifts observed in various places are given in the following paragraphs:

In a limestone mine near Manheim, W. Va., the drifts are 45 feet wide and 26 feet high. The pioneer bench is 14 feet high and extends the full width of the drift; the underlying rock is removed in one 12-foot bench. The pioneer bench is blasted as shown in figure 21. A V-shaped mass, or "core" as it is locally termed, is first blasted out as shown at *x;* the shoulders *y* are next shot out, and then the sides *z*. The 12-foot bench is blasted with a single row of vertical holes, *a*, spaced 6 feet apart and with 10-foot burden. A bedding seam at the floor makes it possible to blast this bench without the aid of "snakehole" shots. The order of blasts is as follows: The vertical bench holes *a* and the two sets of horizontal holes *b* on the shoulders are fired simultaneously; later the side holes *c* and the core holes *d* are fired simultaneously. There is an open bedding seam at the roof, and another 8 to 11 inches below the roof level. Separation takes place more easily along the latter than along the former. Also, this second seam acts as a cushion and prevents shattering of the roof by the shots. As a result the pioneer blasts breaks the bed at this seam, leaving the band of rock 8 to 11 inches thick attached to the roof. However, to leave this thin band up would be unsafe, therefore it is drilled with a stoping machine (vertical drill) and shot down with very light charges. The roof is then very safe, as it consists of a bed of sound limestone 20 feet thick, having very widely spaced vertical seams.

Formerly the bench at the quarry floor was made the pioneer bench, and the upper part was shot down later. It is said that the method gave good results in blasting, but involved dangers from roof falls; also, the expense for cleaning the roof was high.

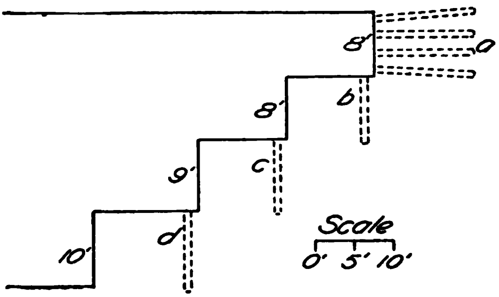

FIGURE 22.—Method of blasting face in drift in a limestone mine in western Missouri. Vertical section lengthwise of drift is shown. *a*, Blast holes for pioneer bench; *b*, *c*, and *d*, holes for second, third, and fourth benches.

In one eastern Missouri quarry, mining was practiced a number of years ago but was later abandoned. The drifts were 45 feet wide and 30 feet high. The pioneer bench was 8 to 10 feet in height, and the underlying rock was removed as a single bench by a blast in vertical drill holes. The shale floor provided a smooth parting and made removal of the entire bench possible without the aid of "snake holes." The pioneer bench and the second bench were blasted simultaneously, the former being kept one step ahead of the latter.

In a western Missouri mine, the drifts are 45 to 60 feet wide and 35 feet high and are worked in four benches. The pioneer bench is 8 feet high and is advanced horizontally 10 to 12 feet with each successive blast. The other three benches are 8, 9, and 10 feet, respectively, in height, as shown in figure 22. Blasts *a* and *b* are fired at one time, and blasts *c* and *d* at one time. The drill holes for the pioneer bench are arranged in three groups as designated by *a*, *b*, and *c*, figure 23. The charges consist of "40 per cent" dynamite and are fired by means of fuses attached to detonators. The fuses for the holes *a* (fig. 23) are shorter than those for the holes *b*, and the latter are shorter than those for the holes *c*. As a result, the blasts in holes *a* are fired first and force out the central V-shaped mass; following this the blasts in holes *b* remove the shoulders, and finally, the blasts in holes *c* throw down the side masses. Each succeeding bench is shattered by a blast in a single row of vertical holes, as shown at *b*, *c*, and *d*, figure 22.

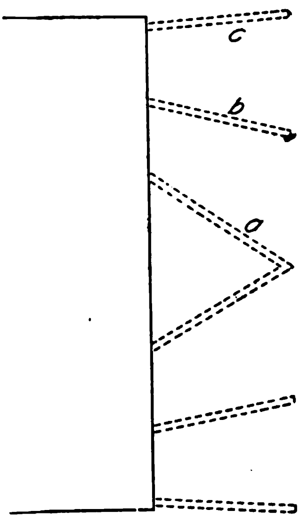

FIGURE 23.—Plan arrangement of blast holes in pioneer bench: *a*, *b*, and *c* are rows of drill holes.

The holes for the blasts *a* (fig. 23), which force out the V-shaped central mass, are known as the "cut holes." To be effective, blasts in the cut holes should be as nearly simultaneous as possible. When fuse is used each pair of holes should so nearly meet at the bottom that the first charge to detonate will detonate the charge in the adjacent hole and thus insure almost simultaneous action. Shots when fired with electric detonators are more nearly simultaneous and more effective than with fuse.

The blasts on the pioneer bench leave a mass of rock about 1 foot thick at the roof. Originally this band was left up, but later was found to be unsafe because a thin layer of shale separated it from the next overlying bed. Of late years it has all been removed and the roof is now in very safe condition.

A unique method of drift mining is practiced in a limestone mine in southern Ohio. The bed of serviceable stone, which is only 8 feet thick, is overlain with a thick bed of sandstone, and underlain with 8 inches of soft blue shale, as is shown in figure 24. The limestone is undercut in the blue shale with a coal-cutting machine, which cuts the shale rapidly, and is then blasted. The sandstone makes a strong roof.

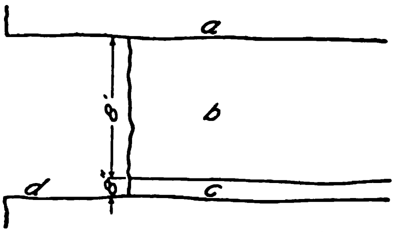

FIGURE 24.—Beds in limestone mine in southern Ohio: *a*, Sandstone roof; *b*, 8-foot limestone bed; *c*, soft blue shale, 8 inches; *d*, floor.

In another limestone mine in southern Ohio the drifts are 15 feet wide and about 22 feet high. The rock is mined in three benches, the pioneer bench being 10 feet high, the middle bench 9 feet, and the lower one 3 feet. The arrangement of drill holes is shown in figure 25. For the pioneer bench there are three series of holes, *a*, *b*, and *c*. The holes *a* are 7½, *b* 6½, and *c* 5½ feet deep. The 9-foot bench is blasted with the four vertical drill holes *d*, and the 3-foot lower bench with the four "drift" or "snake" holes *e*. The five sets of shots are fired in consecutive order at one operation by means of electric detonators.

In blasting the pioneer bench it is obvious that success can be attained only when the three series of shots, *a*, *b*, and *c*, follow each other in order, for the shots *a* must remove the central V-shaped mass before the shots *b* can throw the shoulders in, and the shoulders must be removed before the shots *c* can effectively throw down the sides. Consecutive shots are fired in some quarries by means of fuses of different lengths. However, in this quarry delay-action electric detonators are used. In a detonator of this type a short piece of fuse is ignited by the electric current, and the cap is fired by the fuse. The period of delay is governed by the length of the fuse. The charges *a*, *d*, and *e* are fired by instantaneous detonators, charges *b* by "first delay," and charges *c* by "second delay" detonators.

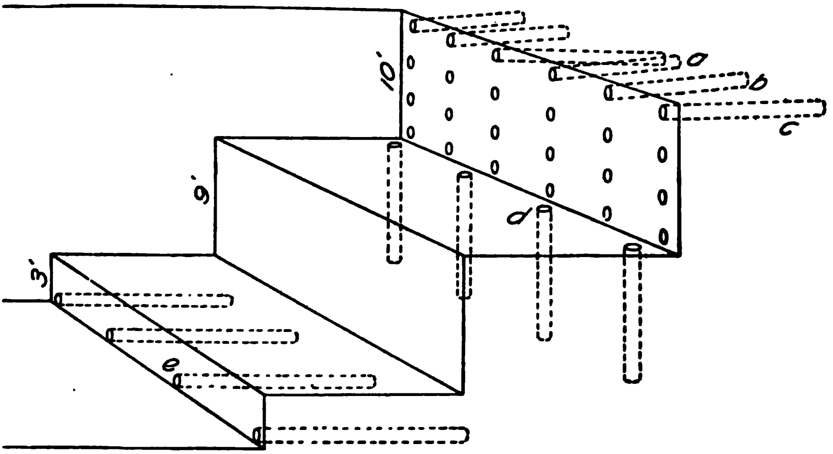

FIGURE 25.—Method of blasting face of drift in limestone mine in southern Ohio: *a*, *b*, and *c*, three sets of blast holes for pioneer bench; *d*, holes for second bench; *e*, holes for third bench.

In an Illinois limestone mine the drifts are 21 feet high and 20 to 40 feet wide, depending on the strength of the roof. The limestone is removed in three benches of 7 feet each. Contrary to conditions in most mines, the middle bench is chosen for the pioneer bench. At the bottom of this bench is a clay seam which makes a plane of easy separation from the lower bench. A side view of the drift is shown in figure 26. The pioneer bench *a* is removed by three rounds of shots in horizontal holes, throwing out first a central core, then the shoulders, and finally the sides, as in the methods described in preceding paragraphs. The upper and lower benches are then shot down with blasts in the vertical drill holes *b* and *c*.

The rock is loaded with electric shovels and compressed-air shovels, and is conveyed to the crusher with electric trolleys.

In a shale mine in Tennessee, operated in connection with an open-pit limestone quarry, the main entry, the air course, the drift necks, and the crosscuts are 8 by 8 feet. The drifts, or chambers, from which most of the production is obtained are 20 feet wide and 16 feet high. The rock in the chambers is shot down with three rows of holes, the bottom row having eight holes and the middle and upper rows three holes each. For the 8 by 8 foot work the lower row consists of four holes, and the upper rows of two holes each. The shattered rock at the walls is cleaned down with picks. Timbering is necessary, as the shale roof is weak and insecure.

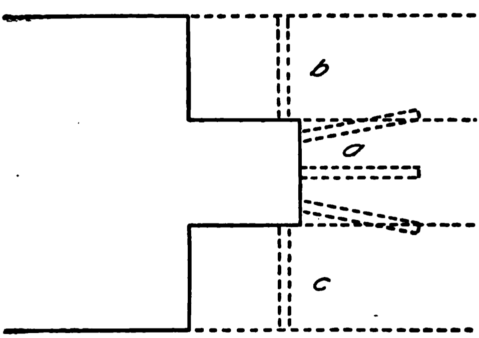

FIGURE 26.—Method of blasting face of drift in Illinois limestone mine: *a*, Pioneer bench, *b*, upper bench; *c*, lower bench; dotted lines represent drill holes.

STOPING FOR LIMESTONE.

What is known as a "shrinkage stoping" method was at one time employed in an eastern Missouri limestone mine. The rock was blasted down by overhand stoping, the working space being kept open by drawing off part of the broken rock from an opening beneath. The serviceable limestone in this mine is 35 feet thick, and is underlain with shale. A preliminary drift, or "heading," was run in the shale; both walls and roof were timbered; at suitable intervals raises or chutes were started in the roof of the heading and these enlarged into stopes, the walls of the stope diverging upward as shown in figure 27. Each stope was about 50 feet wide at the top and 150 feet long. The stopes were separated by "stop pillars," or masses of solid rock, to prevent loose rock from falling from one

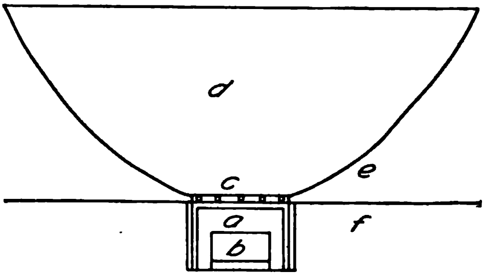

FIGURE 27.—Method of "shrinkage stoping" formerly employed in a limestone mine in eastern Missouri: *a*, Heading; *b*, mine car; *c*, chute; *d*, stope; *e*, limestone; *f*, shale.

stope into another. The broken rock was drawn off, as desired, into cars through chutes in the timbered roof of the heading. The original intent was that branch headings should be made and a number of stopes worked simultaneously; but before this stage was reached the method was abandoned. The plan was, however, scarcely carried far enough for a fair test. The method probably would be more successful when employed in a limestone bed more than 35 feet thick, as a larger tonnage of rock would be obtained for a given expense of driving and timbering a heading.

GLORY-HOLE METHOD OF QUARRYING.

The so-called "glory-hole" method has been employed to some extent in mountainous regions, particularly in West Virginia and Washington. A tunnel is driven into the ledge near its base at a point conveniently situated for transporting the rock to mill or railway. At the inner end of the tunnel a large chamber is blasted and is used as a storage bin for the rock to be removed from above. A slightly inclined shaft is then driven up to the surface. The rock is quarried around this opening in a circle of ever-increasing size, the hole assuming the shape of a funnel. At the bottom of the shaft the flow of rock into the chamber is regulated by segmental gates. The method resembles somewhat the stoping method described in the preceding paragraphs, except that the workings are open to the surface. In one quarry the lower opening is directly adjacent to and above the crusher house and the quarried stone drops through a slightly inclined chute into hoppers, which feed the crusher directly. The glory-hole method of quarrying was first developed in Germany. Its chief advantage is that gravity does the work of a steam shovel.

CARE OF THE MINE ROOF.

The amount of attention that should be given to a mine roof depends greatly on the nature of the rock that constitutes it. A thin bed of rock with a distinct plane of separation between it and the overlying bed makes an extremely dangerous roof. In one Missouri mine a man was killed a few years ago by a fall of rock from such a roof. In a limestone mine with a sandstone roof the presence of masses of limestone clinging to the roof is dangerous, as there is usually a plane of easy separation between the two types of rock. Bedding planes or joints in close proximity tend to weaken the roof, and careful attention should be given to discover signs of weakness, and all loose scale should be promptly removed.

Some companies have a systematic examination of the roof made at specified times. In a West Virginia mine a derrick mounted on a truck is employed for this purpose. The derrick is of light, though strong, construction and is so designed that it may be lowered to pass under electric wires. Working from positions at the top of this derrick one or two men sound the roof and bar down all loose scale.

In one Tennessee mine the shale roof requires timbering to render it safe.

Where the mine entrance is in a steep hillside material dislodged from above may be a source of danger at the entrance. A noteworthy safety device observed by the writer is a timber roof extending from the face of the hill to a point 15 or 20 feet outside the main entry. Its purpose is to prevent falling débris from injuring workmen as they enter or leave the mine.

ROCK LOADING IN MINES.

Power shovels operated by compressed air are in use in mines near Independence, Mo., Oglesby, Ill., and Lowmoor, Va. In the former locality two shovels of the tractor type are employed to load rock into side-dump cars of 4-ton capacity.

In most limestone mines the breast is too small and work conducted in too many places to make power-shovel loading practicable. Hand loading is, therefore, the common method in such mines. Where the drift is wide, as at Manheim, W. Va., cars are placed on two tracks, one near each side of the drift. With every convenience provided, hand loading is usually less efficient than power-shovel work, and this factor has a distinct bearing on the high cost of rock-mining operations.

To insure the greatest degree of safety, benches should not be cleaned at the same time as loading. When for any reason hand loading is temporarily suspended this interval should be employed for bench cleaning.

ROCK TRANSPORTATION IN MINES.

In the shaft mine in Ohio referred to previously, cars of 2½-ton capacity are hauled to the main hoist with mules. The cars are then run into the cage and hoisted to the surface. Mule haulage is also employed in mines having horizontal entries. In one mine substitution of storage-battery locomotives for mules is contemplated.

An electric motor is employed for haulage in a southern Ohio mine. As the drifts are in places less than 6 feet in height, an objection to the use of a trolley is the danger of contact with trolley wires where there is not sufficient head room.

In one Missouri quarry, the loaded cars are hauled by horses to the main haulage way and brought to the mine entrance by locomotive. At the present stage of mining, the drifts are short, and surface openings at several points provide adequate ventilation, even where steam locomotives are employed.

In a West Virginia mine gasoline locomotives are used. Three locomotives are employed, two being in active service and one held in reserve. The chief objection to their use is that they may be easily deranged and thus render unreliable service. Those who operate or contemplate operating gasoline locomotives in a mine should carefully consult the various published articles on this subject.[a] Efficiency and safety in the use of gasoline locomotives depend on the proper choice of a machine and on careful observance of the rules governing its operation.

ROCK TRANSPORTATION FROM MINES TO CEMENT PLANTS.

At shaft mines the loaded cars may be run directly from the hoisting cage to the plant by cable haulage, if the distance is short. Where electric motors, gasoline locomotives, or steam locomotives are employed in the mine drifts, it is customary to haul the cars to the crusher by the same means. An aerial tramway is in operation at one Pennsylvania mine. Where the crusher is situated some distance from the plant, the crushed stone may be brought to the plant with cable cars. In one mine the mine cars are run out on a trestle and are dumped into a storage bin, from which the rock is drawn off into hopper-bottom cars of 10 to 12 ton capacity. The latter are conducted to the plant down a steep incline by cable. In a West Virginia mine the entry is near the top of a high and rugged river bluff, and all rock is transported by cable cars down a steep incline many feet in length.

ROCK LOADING.

METHODS EMPLOYED.

After the rock has been stripped, drilled, and blasted the next step is to load the broken fragments ready for transportation to the crushers. As previously stated, two methods of loading are in common use, hand loading and steam-shovel loading. In about five-eighths of the open-pit limestone and cement-rock quarries visited, steam shovels are employed, and in three-eighths of them the rock is loaded by hand. Of the nine limestone or shale mines visited, power-shovel loading is employed in three, and hand loading in six.

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[a] See Hood, O. P., and Kudlich, R. H., Gasoline mine locomotives in relation to safety and health: Bull. 74, Bureau of Mines, 1915, 83 pp.; also Hood, O. P., Gasoline locomotives in relation to the health of miners: Am. Inst. Min. Eng., Bull. 94, October, 1914, pp. 2607–2611, and discussion, Am. Inst. Min. Eng., Bull. 100, April, 1915, pp. 892–894.

HAND LOADING.

In the hand-loading method two men are usually employed at each car to pick up the pieces of rock and throw them into the car. Blasting is so conducted as to reduce the blocks to small size, any further breaking necessary being done by sledging. The fine material is thrown into the cars by means of specially constructed forks.

ADVANTAGES OF HAND LOADING.

When rock is loaded by hand it may be sorted and carefully classified. Where bands of dolomite occur in the rock ledge the fragments of dolomite may be thrown to one side and thus the magnesium content may be kept to a minimum. Where flint bands occur the flinty masses may also be thrown aside. Where the rock contains pockets or seams of clay that can not be easily removed prior to blasting, the clay can be separated from the rock during the loading. Accumulations of earth or inferior rock may be loaded into cars and removed to the dump.

The greatest advantage claimed for the hand-loading method is that it allows great freedom in selecting rock from various parts of the quarry. Usually hand loading is carried on at numerous points simultaneously. If the rock is of variable composition, cars are tagged in such a manner as to indicate the place where they were loaded, and thus the approximate composition of the rock in each car is known. The chemist may then designate the order of dumping the cars and the number of cars of each kind required to give the best mixture.

Hand loading has a decided advantage where a limited amount of capital is available, for the loading equipment is not costly.

DISADVANTAGES OF HAND LOADING.

Breaking rock into small fragments requires an excessive amount of blasting. Thus, the drilling and explosives costs are relatively high where hand loading is employed. Hand sledging, which is employed to supplement blasting, is a slow and costly method of breaking rock. Moreover, hand loading requires many more laborers than are required for steam-shovel work for a given output of rock. This is an important consideration at times when labor is scarce and wages high.

STEAM-SHOVEL LOADING.

Steam shovels have replaced hand loading in a majority of open-pit quarries where rock is obtained for cement manufacture. The shovels are of various types and sizes. Many 60 to 100 ton shovels, designed to operate on tracks and having 1¾ to 4 yard dippers, have been observed by the writer. Others are of the tractor type, that require no tracks. They are smaller than those which operate on tracks, having dippers of ⅞ to 1¾ yards capacity. Tractors give the best service where the quarry floor is firm and smooth. If the quarry floor is parallel with the bedding and coincident with an open bedding plane, a smooth floor is easily maintained and is therefore favorable for tractor-shovel operation or for laying tracks. If the floor is shale or other soft rock, the tractor wheels may require plank supports. For any type of shovel a smooth floor is desirable.

A steam-shovel gang consists of one runner, one craneman, one fireman, and three to six pit men. Some of the smaller tractor shovels require no craneman. The first cost of a steam shovel is high, but with proper care the cost of maintenance is not excessive.

The amount of material handled by a steam shovel per day varies widely, as it depends on efficiency of operation, the conditions of the rock mass, time required to spot cars, and other factors. Tractor shovels are capable of handling 400 to 700 tons of rock per day, while the larger types, with 4-yard dippers, may attain a maximum of 1,400 to 1,600 tons. A fair average for one of the larger types working in limestone is 600 to 1,000 tons per day.

ADVANTAGES OF STEAM-SHOVEL LOADING.

A steam shovel can load blocks of considerable size, hence blasting costs are relatively lower than for hand loading. Moreover, the slow and laborious hand sledging is unnecessary.

A second advantage is the rapidity with which material may be handled. One quarryman reports that a small steam shovel in his quarry loaded 57 6-yard cars of limestone in five hours. Coupled with rapidity of loading is the fact that only a small gang of men is required for a steam shovel, whereas a large gang of hand loaders would be required to handle the same amount of material. This advantage is most keenly felt when wages are high and labor is scarce.

DISADVANTAGES OF STEAM-SHOVEL LOADING.

When rock is loaded by hand it is usually obtained from many parts of the face simultaneously. The steam shovel, however, loads for a considerable time from one position. This is the chief objection to the use of steam shovels in quarries of variable rock, as it does not permit sufficient mixing of the rock from the various parts of the face.

A second disadvantage is the inability of the steam shovel to sort the material that it loads. If clay is thrown down with the rock it must be loaded with the rock. If bands of dolomite or flint occur in the ledge, in general such material must be loaded with the good rock, although with careful manipulation it may be possible to set the larger undesirable masses to one side.

When the sum available for investment in equipment is small, the high first cost of a steam shovel may be an objection to its introduction.

DESIRABILITY OF CHANGING FROM HAND METHODS TO STEAM-SHOVEL LOADING.

At the time of writing, labor is one of the most difficult problems with which quarry operators are confronted, especially in Northern States. Consequently, any equipment designed to give efficient service with a minimum of man power appeals directly to the quarry operator's needs. Therefore, a comparison of the work accomplished per man by each method is desirable.

Figures obtained from 14 quarries where steam shovels are used indicate an average daily tonnage of about 112 tons of rock loaded per man. The men represented in this determination are pit men and steam-shovel men only. For 11 quarries where hand loading is employed, the average daily tonnage per man, loaders only, is 16. This indicates that about seven times as much rock can be loaded per man with steam shovels as can be done by hand. The cost of labor for hand loading by contract is 10 to 12 cents per ton, whereas the operating cost of steam-shovel loading, not including repairs or upkeep, is 4 to 6 cents per ton.

Neither the cost of loading nor the number of tons loaded per man daily are alone a measure of the relative efficiency of the two methods, for other factors enter into the problem. The blasting costs are higher for hand loading than for steam-shovel work, because the rock must be broken into smaller pieces. On the other hand, the cost of crushing hand-loaded material will obviously be lower than that of crushing the massive fragments loaded with steam shovels. Hence, we may assume that these two factors, namely, the higher cost of explosives for hand loading and the higher cost of crushing for steam-shovel loading, will balance each other, although the former figure is probably the higher of the two.

There must also be added to the cost of operating a steam-shovel the interest on the first cost of the equipment, a depreciation charge, and the expense of maintenance. It is evident that, even with the addition of these items, steam-shovel loading under normal conditions is cheaper than hand loading.

SOLUTION OF DIFFICULTIES IN CHANGING FROM HAND LOADING TO STEAM-SHOVEL LOADING.

It has been shown in the preceding paragraph that, in general, the use of steam shovels has decided advantages over the hand-loading method. Hence, any possible solution of the chief difficulties in introducing steam shovels where hand loading is now practiced is to be desired.

Sufficient capital to purchase a steam shovel is, of course, a prerequisite condition.

In some quarries work is conducted on such a small scale that a steam-shovel equipment would be deemed unprofitable. The small tractor shovel is well adapted for small quarries. In general, quarrying can not be done as cheaply on a small scale as on a large scale, and it is doubtful whether cement could be manufactured profitably on a scale so small that the necessary rock could not be handled to advantage with a steam shovel, provided the bulk of the material is obtained from a single quarry.

In several quarries the overburden is shot down and loaded with the rock. Where the overburden varies greatly in thickness from point to point, a proper mix may be obtained by loading at various points simultaneously. In small quarries requiring only one steam shovel this is difficult if not impossible to do, consequently an excess of soil may be loaded at one time and a deficiency at another time. One quarry in Kansas overcame this difficulty by stripping all the soil off with one steam shovel, loading practically pure rock with another shovel, and mixing the clay and the rock proportionally after crushing. Another solution of the difficulty is to have a large storage bin in which the raw materials may be distributed uniformly and thus equalize any irregularities in loading.

Similarly, the presence of clay-filled seams or pockets is made an occasion for resort to hand loading, because of the inability of the steam shovel to sort the material properly. Hand loading of such material is a slow and costly process, and the quarryman is justified in adopting any reasonable expedient to make the steam-shovel method practical. The use of a clam-shell bucket on a derrick arm has been suggested as a means of removing clay from large pockets. If the material can be used as a cement constituent, the use of a large storage bin, as previously mentioned, will tend to equalize the proportion of rock and clay. If the content of the seams and pockets is undesirable, it may be possible to remove such material by installing washing equipment in the screens. The removal of sticky clay by washing would be difficult or impossible, but usually such clay may be used as one of the cement constituents and need not be removed.

The chief objection to the use of a steam shovel in quarries having rock of variable quality is the difficulty of obtaining a uniform and properly proportioned mixture. As a rule, the rock is fairly constant in a given bed, the greater variations occurring in passing from one bed to another. Many quarrymen deem it necessary to blast and load the rock from the various beds separately and later mix it in definite proportions. However, the writer believes that in either flat lying or inclined beds an approximately uniform mixture may be obtained by developing a high face and shooting it down as a single bench.

In some deposits when a high face is shot down as a single bench the rock along the face varies considerably in chemical composition. In one Pennsylvania quarry of this type the steam shovel, by means of a special arrangement of tracks, loads at various points successively and thus supplies any desired rock mixture. Ordinarily a steam shovel works in cuts paralleling the quarry face, but in this quarry the main track parallels the face, and a series of branch lines meet the face perpendicularly, as shown in figure 28. When a sufficient amount of rock has been loaded from one branch line the steam shovel is moved to the main line and shifted to another branch, an operation that requires only a short period of time. The branch lines are so arranged that cars may be loaded on the branch adjacent to the one from which the steam shovel is working.

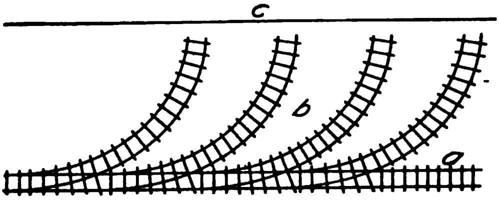

FIGURE 28.—Track arrangement suitable for quarries with high face: *a*, Main line; *b*, branch lines; *c*, quarry face.

In another quarry the tracks are similarly arranged to avoid loss of time in secondary blasting. When large blocks impede steam-shovel work, the shovel is backed to the main line and shifted to another branch, where loading is resumed with the loss of a few minutes only. By this method the shovel is not kept idle while rock masses are being drilled and blasted, but additional service in spotting cars is required. In ordinary steam-shovel practice, a train of cars may be loaded in rapid succession, whereas with the method described above, loaded cars must be switched to the main line and empties returned to the branch singly, a procedure which consumes much time.

As mentioned in a preceding paragraph, the problem of obtaining a uniform mixture is greatly simplified by having ample storage room for raw rock, and by distributing the rock uniformly in the storage bin.

Another difficult in changing from hand loading to steam-shovel loading is the necessity for modifying the car equipment to suit the more strenuous requirements. Hand loading is ordinarily easy on cars and small cars are adequate. When a steam shovel is used larger and stronger cars are necessary. When a steam shovel is introduced it is probable that the purchase of new car equipment would be more satisfactory than any attempt to enlarge or otherwise remodel old cars.

EFFICIENCY IN STEAM-SHOVEL OPERATION.

In changing from hand loading to steam-shovel loading, the small crusher and the small cars commonly used in conjunction with hand loading may make the purchase of a small steam shovel seem advisable. Also, if the rock varies in composition in different parts of the quarry it may be deemed wise to employ two or more small shovels in preference to one large one. In some instances the use of small shovels may be justified, but the operator should, wherever possible, employ shovels of at least 95-ton burden. Rock loading is rough and heavy work, and repair bills for small shovels may be excessive. The employment of large strong shovels involves little expense for repairs or replacements. Moreover, the use of small shovels increases the costs of secondary blasting, as the rock must be broken smaller than for large shovels.

In order to utilize the loading capacity of a steam shovel to its full extent, certain conditions must be met. The chief conditions on which successful operation depends are discussed in the following paragraphs.

The steam shovel should be kept in good repair and should be operated by a skilled runner. An unskilled runner will delay the entire process of loading and transportation.

Rapidity of loading depends largely on successful blasting. Thorough shattering of the rock is essential. If the shovel is repeatedly delayed while large rock masses are being blasted the rate of loading will be slow.

Each cut made with the shovel should be of the maximum width it is capable of handling properly. With each advance of the shovel much time is consumed in moving and laying tracks; hence it is desirable to load a maximum amount of material from each position. When a narrow cut is made the quantity of rock obtained for each advance is smaller than with a full cut; consequently time is sacrificed. The same argument may be advanced with reference to the thickness of the mass of shattered rock to be loaded. Where the mass is thin a small amount of stone is obtained for each move. Moreover, the dipper is filled more easily where a heavy burden provides effective resistance.

Transportation facilities have a direct bearing on the rate of loading. An insufficient number of cars greatly retards the rate. An instance has been observed where a shovel was idle a large part of the time waiting for empty cars. Similar delays may be caused by unskilled handling of cars. Haulage should, if possible, be so conducted that a train of empties is ready to be placed immediately after the loaded cars are removed.

Undue loss of time may be caused by having to move the shovel when a train is partly loaded. If conditions are such that empty cars are not available immediately after the departure of a loaded train, the quarry operator should aim to so regulate the number of cars in the trains that one or more complete trainloads may be obtained from one position of the shovel. Tracks may then be moved and the shovel advanced during the interval when no empty cars are available.

The rate of loading is influenced to some extent by the type of crusher employed. The crusher should be large enough to take any size of block that the steam shovel can conveniently load. Too small a crusher not only delays the steam shovel by its inability to crush the rock as fast as it is loaded, but places on the shovel the additional work of setting aside for secondary blasting and subsequent loading many blocks that could readily be handled by a larger crusher.

ROCK TRANSPORTATION.

USE OF THE TERM.

The subject of rock transportation involves consideration of the means of removal of the rock from the quarry face to the cement plant. It includes the handling of loaded cars and empties.

QUARRY CARS.

TYPES OF QUARRY CARS.

Quarry cars should be designated to suit the nature of the work required of them. For hand loading, they need not have a capacity of more than 2 or 3 tons. They should be low in order to make loading easy. For steam-shovel work, larger and stronger cars are required. Cars of 5 to 10 ton capacity are commonly used, a maximum of 18 tons having been noted. A steam shovel handles many large and heavy blocks, and the cars must be substantially built to withstand the rough usage they receive in loading and unloading. One company uses metal cars of 5 or 6 ton capacity, mounted on two flexible trucks each having four small wheels.

When steam shovels are introduced to take the place of hand loading, it is usually necessary to change the car equipment, as the cars commonly employed in hand loading are too small to give efficient service with a steam shovel. One company found that cars enlarged to increase their capacity were so weak that repair bills became excessive. It is probably better to purchase new cars of suitable size and strength than to attempt remodeling of old cars. Standard hopper-bottom railway cars are employed in some quarries, though cars of smaller size are usually employed.

Various methods of unloading have been observed. Side-dump cars, the body of the car rotating on a pivot or rocking on a semicircular bar, are the most common. A car with one open side is an unusual type observed in one locality. It is so designed that the load is dumped when the car reaches a portion of the track that has one rail elevated. In several quarries the loaded car is placed in a revolving cage and the rock is dumped by a complete revolution of the cage. Cars that dump at the front by depression of the front of the car or elevation of the rear are also used, commonly in connection with a tipple. At one Illinois quarry, if the rock is so placed in the car that it fails to overbalance, the rear of the tipple platform is raised by means of an air piston. A type of car used in an eastern Pennsylvania quarry has a metal body fixed rigidly to a four-wheel truck. The track just beyond the crusher slants upward at an abrupt angle. When the car reaches this point it is suddenly tilted and the rock falls out at the rear, which is open. The crusher is situated beneath the tracks. The car is simple in construction and easy to keep in repair. One difficulty, however, is that frequently blocks too large to pass between the rails are dumped and may occasion considerable delay.

An adequate supply of cars is important. It is claimed that in many quarries shovels are not actually working 40 per cent of the time, the delay being due to lack of track and car facilities. In one quarry observed, loading was done with a 95-ton steam shovel having a 3½-yard bucket, and only eight 5-ton cars were provided. A locomotive hauled 4 or 5 cars at a time, when it could easily have hauled 8 or 10. So few loaded cars were in reserve that when any delay occurred in the quarry the crusher would be idle almost immediately. On the other hand, if the crusher was for any reason delayed, the lack of empty cars caused almost immediate suspension of loading.

MOTIVE POWER FOR CARS.

A gravity system of haulage, where practicable, is the cheapest that can be employed. If quarry conditions are such that movement of loaded cars is mainly down grade, gravity methods may be used to advantage. Horses or mules are commonly used in quarries where the tracks are level or nearly so, and where loading is by hand. Where steam shovels are used the rock cars are too heavy to be handled with animals, and locomotives are generally employed. Locomotives may be of the small type known as "dinkeys," or of standard size. Side-geared locomotives of the Shea or Heister type are recommended for use on heavy grades. The use of electric storage-battery locomotives is contemplated in at least one rock mine for underground work, but their actual operation has not been observed by the writer. Gasoline locomotives are used in at least two rock mines for underground transportation. Haulage by electric trolley is found to be satisfactory and economical in some places. The most serious objection to the use of electric trolleys is the necessity for frequent moving of trolley poles.

A central-control electric three-rail system has been lately introduced in a northern Michigan quarry. The track is divided into sections with an independent current for each section, and thus cars on different sections are subject to independent control. The movements of all cars are controlled from a central tower. The system is somewhat complicated, but is so nearly automatic that few men are required to operate it.

In many quarries the grades are so steep that a cable-and-drum system must be used. Power may be supplied by steam or by electricity.

In one New York quarry a cable belt is driven at a uniform speed, and the cars are moved along the track by clamping them to this cable.

INCLINED RAILWAY.

Where quarries are situated at high elevations the rock may be brought down on inclined railways. If a car is attached to each end of the cable, an empty car ascending while the loaded car descends, the loaded car supplies all the power necessary. Where this system is employed a powerful and reliable brake is necessary if accidents are to be avoided.

In most quarries rock is obtained at a level much lower than the crusher platform, and the loaded cars have to be pulled up an inclined track. Under such conditions, some type of power hoist is necessary. In many places a single track is employed, the loaded car being hauled up by a cable, dumped, and returned under cable control. The power required for hauling cars may be greatly reduced by employing a double-track system and allowing the weight of the empty descending car to assist in elevating the loaded car.

OVERHEAD CABLEWAY.

The use of an overhead cableway supported by towers has been noted. The rock is loaded into skips or pans that are hauled on cars to positions beneath the cable. The pans are then hoisted vertically, attached to the cable, and conveyed to the cement plant. There are two objections to this method. First, the maximum load that can be hoisted is limited to about two or three tons, and therefore material can not be handled rapidly; second, the passage of the loaded pans across the quarry at a great height involves grave danger to the men from falling rock. At one quarry only has this method been observed, and its abandonment is contemplated.

AERIAL TRAMWAY.

The aerial tramway is another type of cable transportation, which is more satisfactory than the overhead cableway. The following information concerning an aerial tramway in operation in Alabama may be of interest:

A $\frac{3}{4}$-inch endless steel cable travels on supports which keep it 15 to 30 feet above the ground. The rock is carried about 8,000 feet, the cable being about 16,000 feet long. It is driven by a 35-horsepower electric motor. The drivewheel, which is about 8 feet in diameter, is provided with automatic clamps which prevent slipping. The buckets have a capacity of about 1,000 pounds each. There are 70 buckets on the cable at one time, 35 full, traveling toward the cement plant, and 35 empty, returning to the quarry. When an empty bucket reaches the quarry it is automatically unclamped from the cable and is suspended by the carrying device from a single-rail circular track, along which it is easily pulled by hand to a position beneath any one of six rock slides from the storage bin, where it is quickly filled. The loaded bucket is then pulled by hand around the track and is automatically clamped to the traveling cable. The cable makes one complete circuit in 36 minutes and delivers approximately 1 ton of rock per minute at the cement plant. The equipment has given satisfactory service for the past six years.

One important advantage of the aerial tramway system, well illustrated in the installation described, is the ease of conveying rock over hilly ground, thus avoiding excessive cost for grading. Where the cable passes over a hill the strain on it may be somewhat increased, but the power consumption is little, if any, larger than where no hill intervenes, for the buckets descending on one side of the hill balance those ascending on the opposite side.

At one Illinois plant, having a capacity of 4,500 barrels of cement a day, all the rock is transported across a river by aerial tramway. The method is satisfactory if the equipment is closely watched and all defects or weaknesses repaired without delay.

The rock for a New York plant is transported by aerial tramway a distance of 1 mile from the top of a high hill. There are 40 buckets on the line, each having a capacity of 1,300 pounds. The maximum capacity of the tramway is about 1,500 tons per day of 24 hours. Coal required at the quarry is carried in the tramway buckets that would otherwise return empty.

At another New York plant the rock is conveyed about 1,500 feet down a steep hill by aerial tramway. The buckets are of 900-pound capacity. The tramway carries, on an average, about 400 tons per day, although its maximum capacity is much greater.

MINE HOIST.

Where rock is obtained from shaft mines an ordinary mine hoist is employed. The loaded cars are usually hauled by horses or mules to the main shaft, placed in a cage, and hoisted to the surface.

QUARRY TRACKAGE.

LEVEL OR MODERATELY INCLINED TRACKS.

Quarry trackage is an important factor in rock transportation. Efficient rock handling depends largely on arrangement of track, elimination of heavy grades, firm roadbed, and uniform gage. As a rule, tracks are standard-gage width, although narrow gage is used in many places. In any event, the gage should be wide enough to insure reasonable stability of the cars in rough places.

An uneven quarry floor may require much grading, and makes tracklaying difficult. The problem of moving and laying track is greatly simplified if the quarry floor is smooth and uniform. In quarrying limestone beds that lie approximately flat, an open bedding plane at the quarry floor is a great advantage, for it provides the smooth floor desired.

The trackage system must be modified to suit conditions. Where steam-shovel loading is employed, the system is usually simple, as loading is conducted at only a limited number of places at one time. The larger the quarry the more complicated will be the track layout, and in quarries where eight or more shovels are used, a block system and tower-signal station may be employed.

Where rock is loaded by hand in large quarries many working places must be provided. A convenient system designed for hand loading in quarries having reasonably high faces with an ample supply of rock is that shown in figure 28 (p. 122). From a main line parallel with the face a series of branch lines extend to the face. One car is placed at the end of each branch, and usually two loaders are employed at each car. Loaded cars are collected by a locomotive from various branches in succession until enough cars to form a train are obtained. Empty cars are distributed to the various branches in a similar way.

For low-faced quarries with a limited supply of rock at each working place, the face is usually very long, and a branch to every working place would require an excessive amount of trackage. Therefore in such quarries the branch tracks usually are parallel with the face, as shown in figure 29. Several cars are placed on each branch and are loaded simultaneously.

In comparing the two methods it may be pointed out that where the branches parallel the face and a number of cars are loaded on each branch, some inconvenience may result from partly loaded cars being nearer to the main line than fully loaded cars. Then all the cars on the branch may be taken to the main line, where the partly loaded cars are cut out and returned to the branch. Where the branches meet the face perpendicularly, conflict of loaded and empty or partly filled cars is avoided. On the other hand, as only one car is placed on each branch, more switching is required to collect a train of loaded cars or to distribute a train of empties. Where the branch lines parallel the face and three, four, or more cars are on each branch, the collection or distribution of cars is a relatively simple matter.

A trackage system employed in an Alabama quarry consists of branch lines meeting the face perpendicularly and connected with a circular main line provided with a crossover dump. The cars are hauled by mules. Formerly the quarry cars were hauled up the incline to the crusher. By means of the crossover dump they are now dumped into cable cars of much larger capacity. This arrangement has increased greatly the haulage capacity of the cable incline and has materially reduced quarry costs and increased the output of rock.

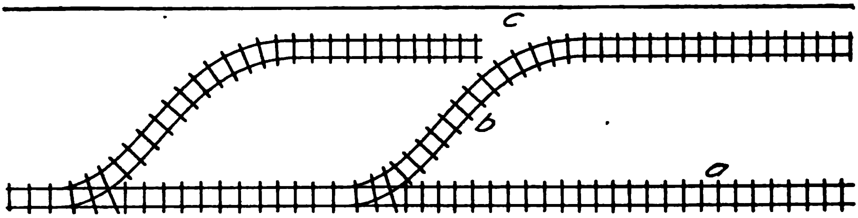

FIGURE 29.—Track arrangement suitable for quarries with low face: *a*, Main line; *b*, branch lines; *c*, quarry face.

Where the hand-loading method is employed in quarries worked on several levels, the trackage is more complicated. An actual example of the complexity in track arrangement that may occur in a quarry worked on four levels is shown in figure 30. The obvious difficulty of transportation under such conditions should encourage the adoption of steam-shovel loading and the working of fewer benches, if circumstances would permit such readjustment.

In many quarries the cars are taken from the quarry by locomotive. Where the quarry has a fixed floor level, this system may be permanent, but where the quarry is gradually deepened the grades may become too steep for locomotives. In some quarries the desire to avoid the expense of introducing a cable incline system has led the operators to retain locomotive haulage on very steep grades. This may make it necessary to purchase additional locomotives, for the trains travel more slowly on heavy grades and are made up of fewer and fewer cars. The installation of a cable system would probably be better than to adhere to locomotive haulage on excessively heavy grades.

Where there is sufficient room for the additional trackage required, a system of back switching may be employed to overcome heavy grades. The system occupies considerable space and involves the first cost and maintenance of relatively long stretches of track, but it permits the handling of a number of cars at a time. In the opinion of some quarrymen, back switching is preferable to cable inclines.

INCLINED TRACKS FOR CABLE CARS.

Where the gradient of the track exceeds that on which locomotives may be operated to advantage, a cable-and-drum system of haulage is usually employed. Single tracks are used in many quarries. In some quarries cars are operated independently on two or more tracks. In others, a car is attached to each end of the cable, one descending as the other ascends, a double track, being provided, or a three-rail track above the center switch and a single track below.

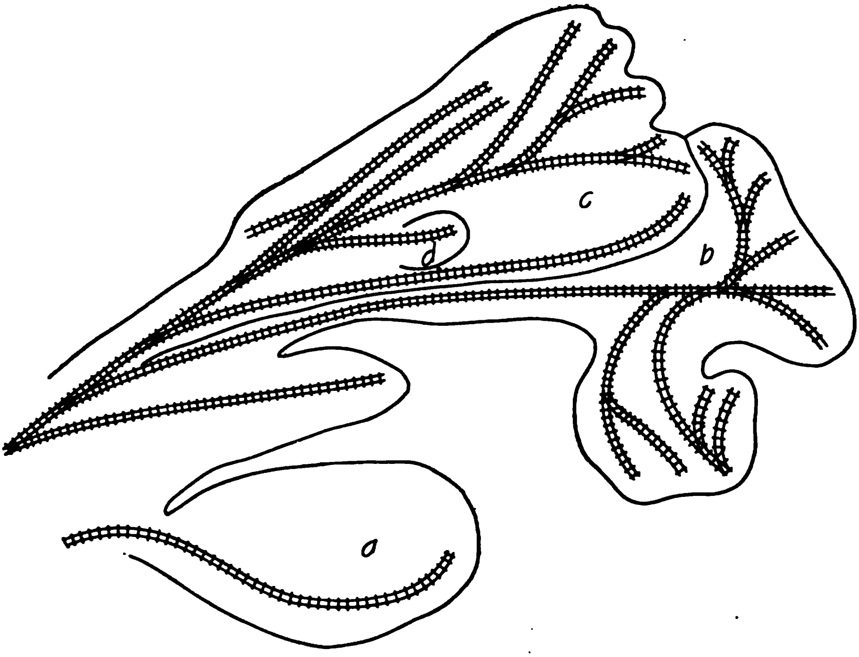

FIGURE 30.—Complex trackage system in a quarry where rock is loaded by hand on four levels: *a*, Upper level; *b*, second level; *c*, third level; *d*, lower level.

On a single track where the load is carried down grade some source of power is required to return the empty car. By the three-rail or double-track system, where two cars are operated by the cable, the descending loaded car pulls up the empty car, and no power is required. Similarly, when transportation is up the incline, the descending empty car helps to draw up the loaded car. The only power required is that needed to raise the actual weight of stone moved, plus the amount consumed by friction, whereas with the single-track system the additional power necessary to move the empty car is required. A considerable saving is, consequently, effected by utilizing the force of gravity, but up to the present time few quarry operators have taken advantage of it.

The "ground hog" or "barney" type of cable haulage is employed in several quarries. A heavy buffer, mounted on four wheels and attached to the cable, operates on a narrow-gage track situated between the rails of the car track. At some distance from the foot of the incline the narrow-gage track runs into a depression below the car-track level. When the cable is out the buffer rests in this depression. The loaded cars pass along the track over this excavation. As the cable winds on the drum the buffer comes up behind the car and pushes it up the incline. Usually one car only is taken up at a time. One advantage of this method is that the inconvenience and danger of attaching the cable to loaded cars and unhooking it from empties is eliminated.

An endless chain system is used as a substitute for a cable in some places. Cars are pulled up the incline by projections arranged at regular intervals on the chain.

TRANSPORTATION FROM POINTS OUTSIDE MAIN QUARRY.

For most cement plants a minor proportion of the raw materials is obtained outside the main quarry. Thus, where limestone and shale are used the plant is usually situated near the limestone quarry, and the shale may be obtained elsewhere. Where cement rock is used, a small amount of high calcium limestone is commonly shipped from outside points.

In some places the shale quarry has its own car tracks and incline, its transportation system being independent of that maintained at the limestone quarry.

Where raw materials are thus obtained from two sources, the tracks and storage bins should be so arranged as to avoid excessive rehandling. A convenient means of handling rock brought from a distance was noted at an Oklahoma plant. The rock is conveyed in hopper-bottom cars and dumped into a trough that opens on a belt conveyor which carries it to storage.

An example of very inefficient handling of material was noted at one Pennsylvania plant. The cement rock, which was quarried close to the plant, required the addition of a small amount of high calcium limestone brought from a distance. The limestone was carried in hopper-bottom cars to a trestle above the quarry pit, which at this point was 40 or 50 feet deep. It was dumped into the pit, loaded into quarry cars by hand, and hauled by mule to the incline up which the cement-rock cars were taken. The cars loaded with limestone were elevated at such times as they were needed to maintain the desired mixture. It would seem that the maintenance at a high level of a limestone storage bin from which the rock could be drawn by gravity or conveyor would add greatly to the efficiency of this transportation system.

TPYICAL TRANSPORTATION SYSTEMS.

NECESSITY FOR COMBINATION OF TWO OR MORE METHODS.

In order to avoid the extra cost of rehandling cars an effort should be made to simplify the transportation system as much as possible. At most cement plants, however, the rock must be elevated from the quarry to the bank, from ground level to an elevated crusher, or brought from a high level to a lower level. This necessitates the employment of cable inclines for the hills or trestles, and thus involves a combination of two or more methods of transportation.

SINGLE METHODS OF TRANSPORTATION.

In a few localities the simplest system of transportation is possible, namely, the transfer of rock from the quarry face to the crusher by a single process. Thus, at New Castle, Pa.; Hannibal, Mo.; and Superior, Ohio, the crusher is on a hillside. The rock is quarried at higher levels and is brought to the crusher directly by locomotive or electric trolley.

COMMON COMBINATIONS OF METHODS.

The combination of methods most commonly observed is haulage of cars in trains by locomotives to the incline and cable control up or down the incline, as the case may be. In many quarries the cars are hauled to the incline with horses or mules.

In five quarries, having a gentle down grade from the face to the incline, the loaded cars are let down from the face to the foot of the incline by gravity. The empty cars are usually returned by horses or mules. Where the grade is light and the cars are small they may be pushed to the face by the men.

SPECIAL COMBINATIONS OF METHODS.

In a deep Pennsylvania quarry with somewhat limited floor space, loaded cars are hauled from the point of loading to the incline by means of a cable operated on a drum attached to the steam shovel. A cable-and-drum equipment, operated by electricity, hauls them up the incline.

A gravity method, where practicable, is the cheapest possible power that quarrymen can utilize. A description, therefore, of two systems where gravity is employed effectively may be of value to the industry.

In one Pennsylvania quarry the loaded cars run on a down grade to the cement plant, where they are elevated, dumped, and returned by gravity on a trestle which descends on a gentle grade to a point near the quarry face. Thus gravity transports both loaded and empty cars.

In a quarry in western New Jersey both loaded and empty cars are moved by gravity to the foot of the incline. The trackage system is shown in figure 31. The incline has a double track, the empty car descending as the loaded car ascends. Thus the weight of the empty car assists in elevating the loaded car. The empty cars leave the incline 30 or 40 feet from the bottom, at *c*, and run on a gentle down grade to *d*, where the track is elevated abruptly. This elevation stops the car, and as it returns an automatic switch directs it to the track leading to the steam shovel, *e*. The car is thus turned around. It descends by gravity to the steam shovel, where it is stopped by placing a block on the rail. When the car is loaded the block is removed and the car again travels by gravity to the foot of the incline, *f*, where the hoist cable is attached. The success of this system depends chiefly on getting exactly the proper grade, which must, of course, be governed by the ease of running of the cars. If the grade is too gentle the cars may stop, and if it is too steep they may be stopped with difficulty or may jump the track. This method eliminates all

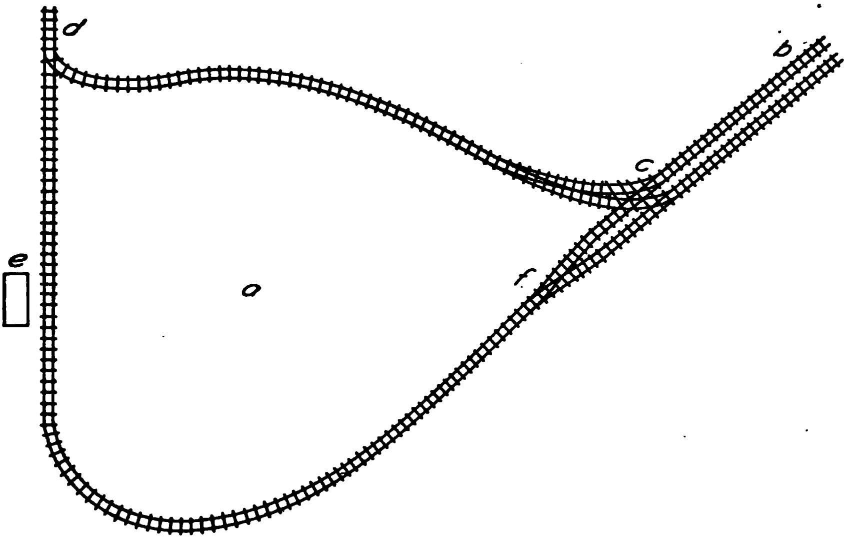

FIGURE 31.—An efficient track system used in a western New Jersey quarry: *a*, Pit; *b*, incline; *c*, point at which empties are switched off; *d*, steep incline where empties are stopped and reversed; *e*, steam shovel; *f*, foot of incline.

horses, mules, and dinkey engines from the quarry. The system is so nearly automatic that little labor is required. With a gang of only 16 men this quarry can produce about 1,200 tons of rock per day.

IMPORTANT FACTORS IN SUCCESSFUL TRANSPORTATION.

In order to attain high efficiency in transportation, the quarryman should aim to have strong cars, an ample supply of cars, a simple haulage system, a combination of as few methods of transportation as possible, and should utilize the force of gravity wherever practicable.

ROCK CRUSHING.

LIMITS OF DISCUSSION.

Rock crushing is a process of manufacture rather than a quarry process, consequently no attempt is made to discuss it herein except insofar as it influences or is influenced by quarry methods or rock structures. The features of special interest to quarrymen are size of crushers, rate of crushing, and situation of crushing plant with respect to rock transportation.

SIZE OF CRUSHERS.

The size of the crusher should be governed by the size of the rock fragments that can be loaded and transported. Thus, when hand loading is employed the small fragments obtained may be easily handled by a small crusher. In most quarries where the hand-loading method is employed a gyratory crusher of about No. 9 size is used. Where loading is with steam shovels larger crushers are desirable. Gyratory crushers of No. 18 and No. 21 sizes have been observed where steam shovels with 3½-yard or 4-yard dippers are employed.

The attempt to use a small crusher for breaking rock loaded with a steam shovel is likely to result in great loss of time. Many blocks that could easily be loaded with the shovel must be set aside and broken by blasting, because they are too large for the crusher. Also, if the crusher is too small, the larger pieces may jam the crusher and cause much delay.

At one quarry where rock loaded with a steam shovel was crushed with a No. 9 gyratory crusher, as much explosive was used for secondary as for primary blasting because the rock had to be broken into such small pieces.

The size of the crusher required depends to some extent on the character of the rock cleavage. Some rocks have "cubic cleavage," that is, the rock tends to break into cubical blocks. Others have a "slab cleavage," and tend to split out in relatively thin slabs. Thus, at one cement rock quarry the ratio of length to width to thickness of the average fragment is 5 to 3 to 1. A cubic block 2½ feet in size is approximately of the same volume as a slab 5 feet long, 3 feet wide, and 1 foot thick. The slab would pass through a rectangular opening only 3 square feet in area, whereas the cubic block would require an opening more than 6 square feet in area. Obviously, for crushing a given volume of stone per day, a larger crusher should be provided for rock having cubic than for rock having slab cleavage. The following comparison of data may serve to illustrate the effect of rock cleavage on ease of crushing.

At one quarry where the rock has slab cleavage, the rock was loaded with a No. 61 Marion steam shovel and crushed with a No. 9 gyratory crusher. Three men were employed to dump the rock cars and conduct the rock into the crusher, and the average daily output was about 700 tons. At another quarry where the rock has cubic cleavage, the fragments were loaded with a No. 60 Marion steam shovel, and crushed with a No. 9 gyratory. crusher. Four men were employed to force the rock into the crusher, and the daily output was about 400 tons. It will be noted that in the quarry where the rock broke in cubic blocks a smaller shovel was used, and consequently smaller fragments were loaded. However, notwithstanding this advantage, more men were required to force the rock into the crusher, and a much smaller daily output was obtained.

CAPACITY OF CRUSHER.

Where steam shovels are employed, the crusher should be large enough to handle any blocks that can be conveniently loaded, and also should be able to crush with sufficient rapidity to prevent delay in loading and transportation. Where gyratory crushers are used, slippery clay mixed with the rock may greatly decrease the capacity of the crusher during rainy periods. The effect of clay is less pronounced in large gyratory crushers. Large rolls provided with blunt teeth are used for crushing at several plants and give satisfactory service where the rock is not exceptionally hard. They may be operated in pairs, the rock being crushed between them, or singly, a baffle plate being substituted for the second roll. They have the advantage of having wide, hopper-like mouths that will take large-sized fragments. There is little danger of delay through jamming of rock fragments in roll crushers, although the rock may arch in the hopper if a large quantity is dumped in at one time. The larger sizes of rolls are capable of crushing rock very rapidly. A single roll 60 inches in diameter and 84 inches long was observed working in limestone that was loaded with steam shovels. When timed by the writer, it crushed the rock to 8-inch size or smaller at the rate of 11 tons per minute.

Jaw crushers are preferred by a number of quarrymen. On account of the wide mouth little labor is required in feeding the rock into the crusher, also the cost of secondary blasting is reduced. At one jaw crusher observed, a traveling apron governs the supply of rock and equalizes the flow, which would otherwise fluctuate greatly when large carloads are dumped into the hopper at intervals.

DEVICES FOR MOVING JAMMED BLOCKS.

Loosening blocks of stone jammed in a crusher by using a bar or sledge may delay and even temporarily suspend quarry production. Most well-equipped crushing plants are provided with mechanically operated hooks, by means of which a jammed block may be readily moved. In some plants the hook is raised with a rope tackle or with some form of cable hoist. The pneumatic hoist is probably the quickest and most convenient, but can be used only where compressed air is readily available.

SITUATION OF CRUSHER.

The method of transportation is governed to a large extent by the situation of the crushing plant. At a few quarries the crusher is so situated that little or no elevation of stone is required. At one New York quarry the crusher is placed on a hillside, at a lower level than the quarry floor, and the rock is discharged at a still lower level into a storage bin. This simplifies transportation, as no cable-car incline is required, and minimizes power consumption, for at no stage is the rock raised to a higher level. Some such method should be followed wherever possible, but in most places the topography makes elevation of the rock quarried unavoidable.

SAFETY AROUND CRUSHERS.

It is dangerous to work around crushers, especially those of the roll type. Adequate measures should be taken to prevent accident to workmen barring down rock in the crusher hopper. Attention of operators is directed to a safety device employed in an Indiana and a New York quarry. Each man who works close to the crusher mouth is provided with a leather belt to which a rope is attached at the back. The rope is fastened securely at the other end, and is of sufficient length to allow freedom of action, though short enough to prevent the workman from falling into the crusher.

MIXING THE RAW MATERIALS.

MIXING OF RAW MATERIALS IN RELATION TO QUARRYING.

The mixing of the raw materials of Portland cement is a manufacturing and not a quarrying process. However, the method of mixing has a pronounced effect on the quarry methods in many places. So complete is this interdependence that at a number of plants the chemist controls the quarry method to a considerable extent. In the preceding pages the saving that may be effected by employing modern quarry methods has been emphasized. However, the difficulty of getting a proper and constant mixture of raw materials may impose such restrictions on the quarry that these methods can not be used. Consequently the mixing process, insofar as it affects quarry methods, is discussed herein.

IMPORTANCE OF MAINTAINING A PROPER MIXTURE.

Probably no part of the process of manufacturing Portland cement requires more care and vigilance than proportioning the mixture of raw materials. On proper mixing depends to a great extent the quality of the cement and, therefore, the success of any cement manufacturing enterprise. Variations in chemical composition within certain limits are permissible, but every cement company finds that a certain definite ratio of principal constituents gives a superior cement. Hence every reasonable effort should be made to maintain this composition as nearly as possible.

CONSTANT AND VARIABLE ROCK LEDGES.

There are two distinct types of quarries—those having rock of fairly constant composition throughout their full extent, and those in which the rock varies considerably in composition from point to point. With quarries of the first type, the method of quarrying has little or no influence on the mixing process, whereas with those of the second type mixing so depends on quarrying and loading that some cement manufacturers think that a proper mixture can not be made without quarrying the rock selectively from various parts of the excavation.

ADVANTAGES OF ROCK OF UNIFORM COMPOSITION.

In quarrying rock ledges of fairly constant composition the methods used have not been subject to limitations imposed by the cement-plant superintendents. As a consequence the quarry superintendents have been enabled to take advantage of the improved methods that have recently been developed in many limestone quarries where rock is obtained for concrete aggregate or road building. Deep-hole blasting and steam-shovel loading have, in such quarries, largely replaced blasting in tripod holes on shallow benches and loading the rock by hand.

DISADVANTAGES OF ROCK OF VARIABLE COMPOSITION.

In most quarries where the rock varies considerably in composition, the plant chemist insists on a certain number of cars of rock being taken from each bench or each part of the face in definite order. The necessity for picking and choosing the rock here and there excludes the possibility of developing a high face, hurling down great quantities of rock at one time by the deep-hole blasting method, and loading it with a steam shovel. Instead, the tripod or hammer drill must be used, the rock must be worked in benches, car tracks multiplied, and the rock loaded by hand.

A COMPARISON OF QUARRY METHODS.

It has been found by careful investigation in many quarries that the quarry costs per ton of rock obtained are as a general rule much lower where the churn drill and steam shovel are employed than where smaller drills and hand-loading methods are used. This is shown more clearly by figures obtained from various quarries. The average total operating or working cost of delivering rock to the crusher—stripping not being included—for 11 quarries employing the hand-loading selective method was 27.4 cents per ton. The highest cost was 35 cents per ton and the lowest 18 cents. An average of the same costs for eight quarries employing steam-shovel methods was 21 cents per ton. The highest was 32 cents per ton and the lowest 12 cents. As indicated by these figures, the costs may be reduced about 23 per cent by the use of steam shovels. Many cement manufacturers, either through lack of appreciation of the advantages of modern methods or through inability to devise new mixing methods, still adhere to the old ways and are thereby seriously handicapped in attempting to quarry rock as cheaply as it is done in quarries having rock of constant composition.

NEED OF IMPROVED PROCESSES OF MIXING RAW MATERIALS.

Therefore, it is a matter of supreme importance to cement companies operating quarries of variable rock to determine whether the method of mixing the cement constituents can be so modified that the quarryman is not forced to retain slow and costly methods. If such a change can be brought about, quarry costs may be greatly reduced, the output readily increased, and the chemical work materially lessened. As a result of observations made at many plants, the writer is convinced that this can be done. No new and untried methods are suggested, but simply combinations of methods now in successful operation.

The object of this discussion is to show that mixing methods and quarrying methods can be so adjusted that the desired mixture may be obtained without sacrificing quarry efficiency. In other words, the author has attempted to point out methods of securing and maintaining a desirable mixture of cement ingredients from a quarry of variable rock, at the same time employing the churn-drill and steam-shovel method of quarrying.

MIXING THE MATERIALS IN THE QUARRY.

Instead of quarrying the rock selectively and later mixing the various types in definite proportion, it is suggested that the rock ledge be shot down in great masses, irrespective of any incidental mixing that may occur, and the desired mixture obtained by methods described in subsequent pages. The more thoroughly the rock is mixed in the process of quarrying, the easier can the proper mixture be obtained at later stages. A number of quarrymen maintain that the charge of explosive in deep churn-drill holes may be so adjusted that the rock in successive variable beds may be mixed with a fair degree of uniformity as it falls to the quarry floor after blasting. Blasting should, therefore, be so conducted as to bring about this result as fully as possible.

However, in many quarries incidental mixing through blasting can not be relied on to give even an approximately uniform mixture. In several quarries bands of shale interbedded with limestone are shot down and loaded with the limestone. If the shale is in bands of uniform thickness and is thoroughly mixed with the limestone, its presence may not be detrimental; but in some places the shale bands vary greatly in thickness, resulting in undesirable fluctuations in the composition of the quarry product. Also, the composition of the limestone itself may vary considerably from point to point.

If the rock, when thrown down by well-adjusted blasts, still varies considerably in different parts of the quarry, the employment of two or more shovels, each loading a different class of rock and all working simultaneously, will tend to increase the uniformity of the product when all the carloads are dumped into the same bin. This method is employed at Martins Creek, Pa.; Earlham, Iowa; Dewey, Okla.; and a number of other localities. Two shovels are commonly employed at the same time, one in high-calcium and the other in low-calcium stone. If the quarry is too small to justify the use of more than one large shovel, two or more small tractor shovels may give more satisfactory results. However, it is believed that the steps outlined subsequently will insure sufficient uniformity in the mixture even when a single shovel is employed. In one quarry observed a single large shovel was operated successfully in rock that varied from 42 to 90 per cent in its content of calcium carbonate.

MIXING PROCESSES IN CEMENT PLANTS.

THE THREE PROCESSES EMPLOYED.

According to the methods now followed in many plants, the maintenance of a uniform and desirable mixture would be impossible by the method of quarrying outlined in the preceding paragraphs. To clearly understand the reason for such limitations, it is necessary to review briefly some of the methods of mixing the raw materials.

Three methods of mixing, the dry, the semiwet, and the wet, are in common use. The dry method is the most common for mixing raw materials obtained from quarries. The wet method is used almost exclusively in the manufacture of cement from marl, the material being excavated wet and kept wet until the moisture is driven out in the kilns. For the semiwet process the raw materials are at first in a dry state, but at some stage water is added, the subsequent steps being similar to those of the wet process.

DRY PROCESSES.

TYPICAL EXAMPLES OF DRY PROCESSES.

Some typical examples of dry methods may be briefly outlined. In one plant the limestone is crushed in a No. 8 gyratory crusher, and all fragments larger than 2-inch are recrushed in a No. 5 crusher. The rock is dried and then passed through a ball mill, after which clay is added. The clay is ground, dried, and added to the limestone in the desired proportion by weight. The resulting mixture is pulverized in Fuller mills. It is sampled every hour, and the proportion of clay needed is calculated from analyses of the samples.

At another plant observed, the crushed rock is conveyed to a storage bin, after which it is passed through driers and then through hammer mills. Shale is added and the mixture is pulverized in Fuller mills and is then ready for burning. As before, analyses of the mixture are made every hour.

At a third plant, the crushed and dried rock is passed through ball mills, the desired amount of shale added, the mixture pulverized in tube mills, and then burned.

A noteworthy feature of the methods outlined, which is common to most of the dry-mixing methods observed, is the fact that the proportions of the raw materials are usually fixed by analyses of preceding mixtures, and that there is no means of changing the mixture after it is once made, except insofar as a change in subsequent mixtures may affect it. In other words, one mixture is analyzed, and as a result of this analysis a change is made in a subsequent mixture, which may or may not need the change. If the raw materials are subject to only slight fluctuations in composition, the method gives satisfactory results, but if marked changes in composition occur an undesirable mixture is likely to result. The selective, hand-loading method of quarrying is the direct outcome of this inability of the plant chemist to cope with marked fluctuations in composition of the raw materials. In some plants, mixing has become a process of almost pure guesswork. The chemist becomes very familiar with the rock, judges its composition with fair accuracy from its appearance, and regulates the mixture on the basis of this judgment. Needless to say, the method demands constant vigilance and careful quarry supervision.

METHODS OF OVERCOMING FLUCTUATIONS.

One fairly effective method of meeting the difficulty is to run the mixed and pulverized material into a large storage bin before it enters the kilns. If a supply sufficient for eight or nine hours' run is maintained the operator may compensate for any undesirable variation. Thus, if the mixture is found to be too high in calcium, an excess of clay or shale may be added for a time sufficient to reduce the average calcium content to the desired point. This method can succeed only where proper mixing of the materials may be brought about in the bin.

A more accurate method of mixing is followed by one Oklahoma plant where the limestone is sampled before the shale is added. The limestone is crushed, dried, and ground, and then by means of an intermittent sampler a sample is taken about every 65 seconds. All the samples for a given period of time are mixed, an average analysis obtained, and the proper proportion of shale calculated and added by weight. As clay stripping is shot down and loaded with the rock in the quarry, a small amount of shale only is added. The mixture is then pulverized in tube mills. The product of five tube mills is all conducted to a single bin, from which it is distributed to the kilns. Such mixing gives greater uniformity to the product.

THE BEST METHODS OBSERVED.

An admirable method observed of making a dry mix is successfully used by a company in the Middle West. Blasting in churn-drill holes and steam-shovel loading are employed. A clay overburden of good quality is shot down and loaded with the rock, and on this account the addition of shale is unnecessary. The rock is crushed, dried, and ground in a ball mill to 14-mesh size. The ground material is then taken to storage bins, and during transit a continuous sampler takes a representative sample, which is analyzed every 3 hours. A record is kept of the rate of entry, the number of hours that rock is discharged into the bin, and the average composition for each 3-hour period. From this data a fairly accurate analysis of the entire contents of the bin may be deduced. The ratio of silica and alumina to calcium is calculated by Newberry's formula that carbonate of lime$=$silica$\times 5+$alumina$\times 2$. For example, if the estimated analysis of the entire bin shows 12 per cent silica, 7 per cent alumina, and 72 per cent calcium carbonate, the desired percentage of calcium carbonate would be 74. As only 72 per cent calcium carbonate is present, the contents of this bin would be expressed as —2. Twelve such bins are maintained, and each is designated by a plus or minus figure, which indicates the excess or shortage of calcium carbonate.

In drawing off the material it is taken from not less than three bins at once, and a combination of bins is selected such that the + and — quantities will give a zero result. The first run is sampled and its composition approved before the mixture is allowed to go to the tube mills for final grinding. This method has advantages over methods of mixing pure shale and limestone. In the latter methods temporary obstruction of the flow of shale or limestone decidedly affects the composition of the mixture, whereas with the method outlined, the contents of the various bins differ so little in composition that the effect of obstructing any one outlet for a brief period would be small. For plants where shale is mixed with limestone, it is, therefore, desirable to add the shale before the material enters the storage bins.

An improved type of bin, having for its object more complete mixing of the materials, was observed by the writer at an Illinois plant. Four bins are provided for storing rock after it is crushed, dried, and ground to 20 mesh. Each bin is divided into four parts by vertical partitions. The parts are filled in succession, but the material is drawn off from all four simultaneously. This tends to equalize any fluctuations in composition. Uniformity in the final mixture is further promoted by having sufficient storage room at the kilns for 3 or 4 hours' supply.

Another improvement suggested is a simple device for proportioning the amount of material from each bin. It consists of a series of sprocket-driven conveyors, each having three speeds. Thus, if material is to be taken from three bins in the proportion of 1 to 2 to 3, the conveyor for the first bin could be operated at slow speed, the second at intermediate, and the third at high speed.

Another company quarrying limestone interbedded with shale maintains a two weeks' supply of rock. The proportions of shale and limestone in the ledge are such that if the entire face is thrown down as a single bench and the mass thoroughly mixed, the resulting mixture has approximately the proper composition for cement. The large storage bin assists in obtaining a uniform mixture. The rock is drawn from the storage bin, pulverized, and conducted to 16 concrete tanks. Continuous samples are taken, the samples being analyzed every hour. From the data thus obtained the average analysis of the material in each tank is determined. The material is drawn from one, two, three, or a maximum of four openings in each tank, and a combination of tanks is determined that will give the desired mixture.

One New York plant maintains six storage bins. The material may be drawn from each bin by two worm feeds, each controlled by a 3-speed cone pulley. Thus six rates of removal from each bin are possible.

Meade[a] suggests that the raw material, after it is passed through ball mills, be stored in tanks, the composition of the contents of each tank being slightly higher in lime than is desired. The material is sampled as it enters the tank and an analysis calculated for the entire mass. It is then an easy matter to calculate the small proportion of clay or shale that must be added to bring the mixture to the proper composition. At least three bins should be maintained for the mixture and one for shale.

A slight modification of this method is proposed by an Illinois cement company. The chemist of this company has found that the composition of the cinders from the boiler room is approximately the same as of the shale used in the cement mixture. It has been proposed, therefore, to run the mixture a little higher in calcium than is desired for cement, and to correct the mixture by means of an auxiliary cinder feed. By this means any fluctuation in composition could be corrected immediately, whereas by the method used at the time of the writer's visit the correction was made about 24 hours after the defective mixture has gone by.

THE SEMIWET PROCESS.

DESCRIPTION OF THE PROCESS.

Fluctuations in the composition of raw materials may also be overcome successfully by use of the semiwet process, also called the semidry process. Both terms are misnomers, as the materials are just as wet as in the wet process. The term "semiwet" implies the subsequent wetting of materials originally dry. The advantages claimed for the semiwet process are intimate mixture of the particles and ease of obtaining a uniform mixture of the cement constituents. This latter phase of the process will be considered here. The process is best described by reference to a concrete example.

At an Iowa plant where the semiwet process is successfully employed, the raw materials consist of limestone and shale. The limestone is variable in composition, the calcium carbonate content varying from 72 to 92 per cent. The rock is shot down by blasts in churn-drill holes and is loaded with two steam shovels. The quarry operators aim to keep one shovel working in the high-calcium and the other in the low-calcium stone, and thus equalize the product to some extent.

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[a] Meade, R. K., Portland cement, 2d ed., 1911, p. 103.

The rock is crushed and passed through a ball mill, after which ground shale and water are added. The mixture is pulverized wet. The slurry, containing about 40 per cent of water, is conveyed by screw feed through horizontal troughs to storage tanks. Seven tanks are maintained, each having a capacity of about 600 barrels of cement. Samples are taken from the troughs with a continuous sampler and analyzed every hour. The composition of the contents of each tank is determined from these analyses. It is customary in some plants to maintain high-calcium and low-calcium tanks.

The method of making the final mixture is very similar to that employed for the dry process described on page 143. Knowing the analysis of the contents of each tank, the operator can easily work out a combination from two or more tanks that will give the desired composition. The eighth tank is reserved for mixing the slurry drawn from the storage tanks. An air agitator is employed to thoroughly mix the contents of the tank, and when so mixed the composition of the entire contents may be determined from a single analysis.

ADVANTAGES OF THE SEMIWET PROCESS.

When wetted to the consistency of a slurry, a very intimate mixture of the particles may be obtained. It is claimed that grinding or pulverizing the material wet requires at least one-third less power than when ground or pulverized dry, although the counter claim is made that the wet material wears the pulverizing machinery much more rapidly than the dry. Pumping the slurry is a very simple and cheap method of moving it from place to place. Also materials mixed and conveyed in a wet condition make no dust.

The greatest advantage, however, is the ease with which variable raw materials may be mixed together and a final mixture of the desired composition obtained, as outlined above. This feature of the semiwet process recommends it as an alternative method in plants working on materials of variable composition.

DISADVANTAGES OF THE SEMIWET PROCESS.

The greatest difficulty in connection with the semiwet process is getting rid of the 30 or 40 per cent of water that has been added. The process was first employed where the water could be evaporated at small cost, because of cheap natural gas. Where powdered coal is used for fuel, drying is more costly. The difficulty has of late years been overcome with fair success by the use of long kilns. In kilns 160 feet long, so it is claimed, all the moisture is driven out during the passage of the material through the first 35 feet of the kiln. Therefore, the same output may be obtained with slurry in a 160-foot kiln as by the dry process in a 125-foot kiln. It must not be presumed, however, that the entire disadvantage of the semiwet process is the additional first cost and maintenance of long kilns. It is unwise to assume that the heat used in evaporating the water would, with a dry mixture in shorter kilns, be entirely wasted. By means of waste-heat boilers this heat may be saved and utilized.

DESIRABILITY OF PROPER PLANT DESIGN.

The realization that cheaper and more rapid methods of quarrying than the selective hand-loading method are to be desired has led a number of operators to seek some process of mixing raw materials that will permit the use of churn drills and steam shovels. The semiwet process has apparently appealed to several operators as the best means of obtaining a uniform mixture of variable raw materials. As a consequence a number of plants that were originally designed for dry mixing have been remodeled and the necessary equipment added to adapt them to the semiwet process. Several plants of this type differ in some important respects from one properly constructed. Thus in the attempt to retain as much as possible of the old equipment, water may be added at a much later stage than in plants originally designed to operate by the semiwet process. The grinding and pulverizing may be done while the materials are dry, the water being added subsequently. As a result such plants have the disadvantages of both processes, the slower and more costly pulverizing of dry materials, and the high fuel cost of burning a wet mixture. Also, it has been pointed out that long kilns are required in operating a semiwet process to advantage. In remodeling a plant to adapt it for this process, the introduction of new and longer kilns is an expense that few cement-plant operators would care to undertake.

In planning to remodel a plant in order to modify the mixing system in such a way as to encourage more advantageous quarrying, the operator should carefully consider the original design of his plant and the extent of remodeling demanded. Considered from every angle, it is probable that the dry method should be retained in plants designed for a dry process. It is evident that less modification would be required to introduce a system of bins as described on page 143, than is necessary to convert the plant into a properly equipped semiwet plant.

THE STORAGE BIN AS A MIXING DEVICE.

It has been pointed out that variable constituents may be mixed to some extent in the quarry by intelligent deep-hole blasting, and that the quarry products may at a later stage be mixed together as desired, either by the employment of a series of bins, the contents of which may be apportioned in accordance with their composition, using the dry method; or by a similar apportionment of the contents of slurry tanks, using the semiwet process.

Final mixing of the materials may, however, be simplified by having a large storage bin for crushed rock. A mixture of remarkable uniformity has been obtained from a rock ledge of variable composition by this means. However, in order to attain the desired result the bin must be used intelligently. The following example will serve to illustrate the proper use of the storage bin:

One plant has a crushed-rock storage bin of 40,000-ton capacity. The rock is distributed uniformly over one-half the bin by means of a belt conveyor, until this half is filled. It is then drawn off while the other half is being filled. The rock is taken from beneath by two belt conveyors. It falls from the bin through 24 slides that are opened in succession for 5-minute periods. Thus through uniform distribution in entering the bin and equally uniform removal, fluctuations in composition of the rock as it enters the cement plant are very slight, and the proper proportion of shale to be added is easily maintained.

INCOMPLETE MIXING IN STORAGE BIN.

The essential factors in successful mixing are proper distribution of the materials in the bin, and removal of the contents in such a manner as to cause further mixing. At a number of plants observed the rock-storage bin is employed merely for storage purposes and its utility as a mixing device is not realized.

At one plant the crushed rock is distributed in the bin with a clamshell bucket and traveling crane, a type of equipment that gives excellent service when properly used. As observed by the writer, however, the material, instead of being distributed in various parts of the bin and removed by taking bucket loads from different points, was taken from the point where it entered the bin from the crusher, and was conveyed directly to the drier. As the rock was loaded at the quarry with a steam shovel it is obvious that by such direct transference mixing was imperfect. Such procedure is justified only where the rock is constant in composition, otherwise a distribution should be made in the bin and material for the drier selected from the mixed mass.

Where crushed rock is deposited centrally in a storage bin, as by a belt or bucket conveyor, the finer materials tend to remain near the point of entry, and the coarser fragments to roll toward the sides. If clay is quarried with the rock, the clay, being finer, tends to accumulate near the center, and the limestone toward the sides. If the outlet of the bin is also centrally placed, an excess of clay will be obtained when the outlet is first opened, and the proportion of limestone will gradually increase. This undesirable condition has been observed in a number of plants. It is wise, therefore, to select for the outlets positions not directly beneath the point of entry. In the 40,000-ton storage bin referred to in a previous paragraph, the rock is distributed centrally by a belt conveyor, and removed from beneath through two lines of holes situated toward the sides of the bin.

Where dissimilar materials are used, such as limestone with clay that pulverizes to a finely divided condition, the tendency to segregate is marked. Where limestone and shale are employed, this tendency is less pronounced. A uniform mixture is therefore more readily obtained in a storage bin where materials of like physical character are employed, shale with limestone, and clay with marl.

Having thus obtained a fairly uniform mixture in a storage bin, the subsequent processes of mixing and proportioning the cement materials are greatly simplified. A large storage bin is particularly advantageous where the dry process is employed.

Where rock is stored in bins after crushing, it may be dried either before or after storing. With some materials, however, such as clay and some shales, the better plan is to dry first, as the damp materials may pack in the bins and make removal difficult.

The best construction for a storage bin is concrete and steel. The roof trusses may be made to rest directly on the bin walls, and the belt conveyors for bringing in the stone may be carried by the trusses.

SUMMARY OF CONCLUSIONS.

Mixing methods in their relation to quarrying may be briefly summarized as follows:

Churn-drill and steam-shovel methods of quarrying are, in general, to be preferred, and mixing of the quarry materials should be so conducted as to permit their use. Mixing may be done by using a number of storage tanks for approximately proportioned mixtures of ground rock, the contents of the tanks being determined by analyses. The final mix is made by drawing off a calculated proportion from three or more tanks. Thorough mixing of the quarry products may be attained by the intelligent use of a large rock storage bin. The semiwet process is an alternative method of obtaining a uniform and desirable mixture of diverse constituents.

ECONOMIC CONDITIONS.

CONDITIONS GOVERNING PLANT LOCATION.

In choosing a location for a cement plant the presence of an adequate supply of suitable raw materials is not the only consideration. Other conditions of equal importance must be satisfied, or the enterprise will be doomed to failure. The chief factors, other than raw materials, on which success depends are market, transportation, and fuel.

MARKET.

It is unwise to attempt the manufacture of cement in localities where the demand for it is limited. A cement plant should have, first, a strong local market area in which there is practically no competition, and, second, easy access to a larger competitive market area. As cement is a very heavy material, transportation charges are relatively high when compared with its value, and on this account the competitive market area is usually restricted. No excellence in limestone or shale can compensate for a small market.

TRANSPORTATION.

The available market area largely depends on transportation facilities. A high freight rate tends greatly to restrict the field. No plant should be entirely dependent on one railroad line unless the railroad is financially interested with the cement plant. At least two transportation routes should be available, one being preferably a water route.

FUEL

In general practice, 120 pounds of coal dust is required to burn one barrel of cement. If coal is also used for power 200 to 300 pounds are required for each barrel of cement. The fuel cost for an average plant is, therefore, 30 to 40 per cent of the total cost of manufacture. A hydroelectric plant may supply the necessary power to operate the machinery, but fuel is necessary for burning the cement. Hence, a good fuel supply at reasonable prices is a necessity.

RAW MATERIALS.

The success of any cement manufacturing enterprise depends largely on the nature, availability, and extent of the raw materials, particularly in areas where competition is keen. The plants that have raw materials that can be quarried, pulverized, and burned at relatively low cost always have an advantage over competitors. On the other hand, if there is a good local market and cheap fuel, a manufacturer may be justified in building a plant where raw materials are of inferior quality.

No plant should be built having less than 20 years' supply of raw material in sight. Each barrel of cement requires approximately 450 pounds of limestone and 150 pounds of clay or shale. Therefore, a plant making 1,000 barrels daily will use annually about 66,000 tons of limestone and 22,000 tons of clay or shale. This quantity is equivalent to almost 1,000,000 cubic feet of limestone and 250,000 cubic feet of shale. A 1,000-barrel plant should, therefore, have 20,000,000 cubic feet of limestone and 5,000,000 cubic feet of clay or shale available on its property.

EFFICIENCY OF OPERATION.

A cement plant established under the most favorable conditions may fail to yield a profit if operated with poor efficiency. The choosing of a site for the plant need be done but once, and the prospective operator may engage the services of a competent engineer to insure its proper location, but the operation of the plant demands constant attention every day. It is, therefore, supremely important that the cement manufacturer familiarize himself with all modern methods and equipment designed to increase efficiency of operation. Where competition is keen those who keep most closely in touch with each new advance are best enabled to reduce the cost of manufacture, and thus widen their markets.

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Bulletin 168

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

RECOVERY OF ZINC FROM LOW-GRADE AND COMPLEX ORES

BY

DORSEY A. LYON AND OLIVER C. RALSTON

WASHINGTON
GOVERNMENT PRINTING OFFICE
1919

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PREFACE.

At its tenth regular session in 1913 the Utah Legislature provided for the establishment of a metallurgical research department in connection with the State school of mines of the University of Utah. As stated in the act providing for this department (Laws of Utah, 1913, ch. 102, sec. 2, pp. 199–200), the purposes of this research department have been to conduct experiments and research, either alone or in cooperation with the Federal Bureau of Mines and other agencies, with a view of finding ways and methods of profitably treating low-grade ores, of obtaining other information that shall have for its object the benefit of the mining industry and the utilization and conservation of the mineral resources of the State, and to publish and distribute bulletins and articles relating to the department and its work.

This act became effective in July, 1913. In January, 1914, the university made a working agreement with the Federal Bureau of Mines. By the terms of this agreement the work of the metallurgical research department is under the direction of metallurgists of the Bureau of Mines assigned to duty at the university and Salt Lake City. From January, 1914, to July, 1916, D. A. Lyon, metallurgist, was in charge of the work, assisted by O. C. Ralston and other members of the metallurgical staff of the bureau who were on duty at Salt Lake City.

As a part of the cooperative agreement, the University of Utah provides five metallurgical fellowships. The fellowships are awarded to graduates of colleges, preferably of mining schools, who have shown special aptitude for research investigations. Their employment extends over the entire 12 months.

Aside from O. C. Ralston, the experimenters mentioned in this bulletin in connection with the work were holders of these fellowships. Persons desiring further information regarding the cooperation between the university and the bureau are referred to Bulletin 157, Bureau of Mines, pages 9–11.

VAN. H. MANNING,
Director.

RECOVERY OF ZINC FROM LOW-GRADE AND COMPLEX ORES.

By Dorsey A. Lyon and Oliver C. Ralston.

INTRODUCTION.

Volatilization in retorts has been, until recently, the only commercial process of producing spelter, hence the zinc mine operators have had to meet the terms of the zinc smelters in regard to the following requirements: (1) Minimum percentage of zinc the ore must contain; (2) the chemical combination of the zinc in the ore; (3) the proportion present of those elements that interfere with the successful recovery of the zinc when the ore is retorted. In consequence of these requirements, only a comparatively small proportion of the zinc ores mined can be shipped directly to the smelters.

ZINC ORES AND CONCENTRATES SUITED TO RETORT SMELTING.

Under normal conditions the low limit of zinc in ores suited to retort smelting is about 25 per cent for oxidized ores and about 35 per cent for sulphide ores. Consequently the ores or products suited to zinc smelting in retorts may be obtained from—

1. Zinc ores free from interfering elements.
2. Concentrates of zinc ores or of zinc-lead ores.
3. Concentrates of mixed and complex ores, containing gold, silver, lead, copper, zinc, etc.

Table 1, compiled from statistics published in the Mineral Resources of the United States for 1914, shows the principal zinc producing States and the kind of ores mined in them. The zinc produced from ore mined in Missouri, Kansas, Illinois, and Wisconsin comes from sulphide (blende), carbonate, and silicate of zinc; in other words, from class 1. The zinc produced in Utah and in Idaho, Montana, Colorado, and Nevada is largely from classes 2 and 3, or lead-zinc ores, the zinc being separated from the lead when the metallic minerals of the ore are separated from the worthless gangue by some form of mechanical concentration.

TABLE 1.—*Production of spelter in the United States in 1914.*

[Figures for tons are in short tons.]

Source.	Spelter (pounds).				Zinc ore and concentrates.								
					Sulphides.		Carbonates.		Concentrates.		Total tonnage.	Average per cent zinc.	Metal content, tons.
	Total.	From zinc ore.	From lead-zinc ore.	From all other ore.	Tons.	Per cent zinc.	Tons.	Per cent zinc.	Tons.	Per cent zinc.			
Arizona	9,792,337	4,644,952	5,147,385	....	1,231	38.00	76	38.82	12,984	40.50	14,291	40.32	....
Arkansas	....	....	....	....	....	....	....	....	[a]743 [b]1,143	....	1,886	....	608
Colorado	22,571,469	10,010,668	8,882,974	3,737,827	135,705	25.00	114,860	24.34	....	....	251,565	24.70	....
Idaho	42,012,435	6,559,167	35,453,268	....	[c]47,371 [d]7,383	45.69 41.80	....	....	....	....	54,754	45.17	....
Illinois	....	....	....	....	....	....	....	....	[a]16,725	36.50	16,725	36.50	....
Kansas	....	....	....	....	....	....	....	....	[e]21,812 [e]38	57.40 34.20	21,812 38	57.40 34.20	....
Missouri	....	....	....	....	....	....	....	....	[a]189,765 [e]19,648	....	....	58.10 38.00	....
Montana	111,580,544	1,932,695	109,647,849	....	....	....	....	....	....	....	....	....	....
Nevada	12,980,232	5,837,593	7,142,639	....	....	....	....	....	....	....	....	....	....
Utah	15,989,267	2,984,486	12,988,440	[f]16,341	8,485	38.61	2,641	34.40	11,869	44.07	22,995	40.94	....
Wisconsin	....	....	....	....	....	....	....	....	[e]108,535 [e]2,787	....	....	35.05	....

[a] From sphalerite.
[b] From silicate and carbonates.
[c] Sulphide zinc ore and concentrates.
[d] Sulphide lead-zinc ore and concentrates.
[e] Carbonate and silicate.
[f] From lead ore.

As stated previously, and as will be shown later, a large part of the valuable mineral content of the ore is lost in effecting this concentration.

As a result of the other requirements that have to be met in zinc smelting, much zinc is lost in the mining and milling of zinc ore.

ZINC LOSSES IN MINING AND MILLING.

LOSSES IN MINING.

In mining the ore the zinc losses aggregate a large tonnage. These losses are due to several causes; for instance, in the average western lead mine the zinc ore remains as a shell around stopes from which the oxidized lead ores have been taken. In most mines the lateral extent of these shells is unknown, and as they generally carry too little metal to warrant mining at present, they are not removed. Later, the lead-ore stopes are filled with waste and the zinc in these surrounding shells is lost. Figure 1[a] shows the present sources of zinc and the ores not utilized.

LOSSES IN MILLING.

As only a small proportion of the zinc ores mined in the United States contains enough zinc to warrant direct smelting, separation of the valuable minerals from the gangue by some method of mechanical concentration is generally necessary.

For the sake of convenience, the concentrating processes used are considered herein in the following order:

1. Hydromechanical processes.
2. Electromechanical processes.
   (*a*) Electromagnetic,
   (*b*) Electrostatic.
3. Flotation processes.

HYDROMECHANICAL PROCESSES.

In treating zinc ores by hydromechanical processes the object is to obtain a concentrate that, if necessary may afterwards be separated into a zinc product and a lead product, or into a zinc product and a product that may contain lead, copper, gold, and silver, and is sent either to a lead or to a copper blast furnace.

As stated previously, a large proportion of the zinc and lead minerals is often lost in the tailing when any ore mined for its zinc or lead content is treated by hydromechanical concentration. As one of the functions of the Federal Bureau of Mines is to conduct investigations looking to the prevention of mineral waste, the bureau has for some time been conducting investigations in the Wisconsin[a] and the Missouri[b] zinc fields in order to ascertain the causes of the losses in milling zinc ores in those States, and, if possible, to devise improvements in the processes used.

---

[a] Lyon, D. A., and Arentz, S. S., Losses of zinc in mining, milling and smelting: Trans. Am. Inst. Min. Eng., vol. 49, 1915, p. 790.

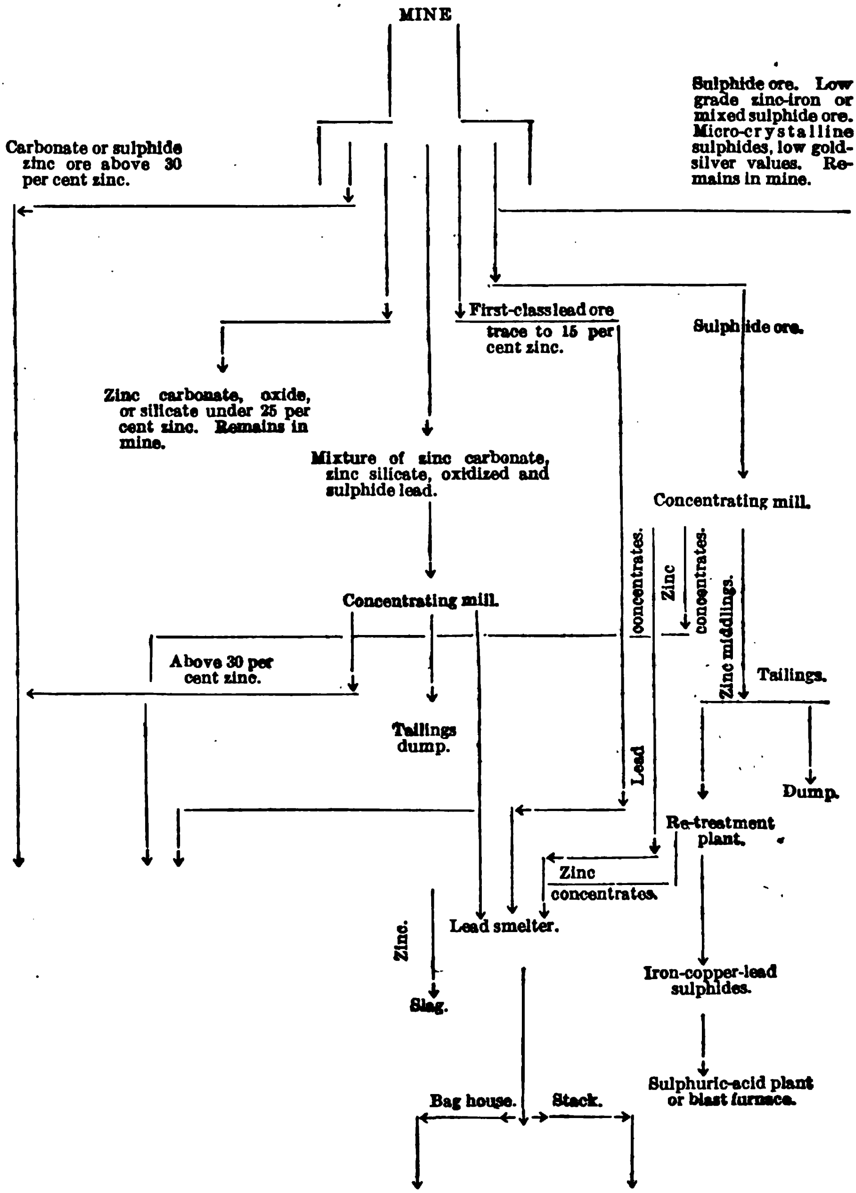
MINE
Carbonate or sulphide zinc ore above 30 per cent zinc.
Sulphide ore. Low grade zinc-iron or mixed sulphide ore. Micro-crystalline sulphides, low gold-silver values. Remains in mine.
First-class lead ore trace to 15 per cent zinc.
Sulphide ore.
Zinc carbonate, oxide, or silicate under 25 per cent zinc. Remains in mine.
Mixture of zinc carbonate, zinc silicate, oxidized and sulphide lead.
Concentrating mill.
Concentrating mill.
Lead concentrates.
Zinc concentrates.
Zinc middlings.
Tailings.
Above 30 per cent zinc.
Tailings dump.
Dump.
Re-treatment plant.
Zinc concentrates.
Lead smelter.
Zinc.
Slag.
Iron-copper-lead sulphides.
Sulphuric-acid plant or blast furnace.
Bag house.
Stack.

FIGURE 1.—Present source of zinc, and also ores not utilized.

---

[a] Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin: Tech. Paper 95, Bureau of Mines, 1915, 39 pp.

[b] Wright, C. A., and Buehler, H. A., Mining and milling of lead and zinc ores in the Missouri-Kansas-Oklahoma zinc district, Bull. 154, Bureau of Mines, 1918, 134 pp.

Wright[a] gives the following interesting data on the efficiency of hydromechanical concentration in the Joplin district. The ores require the simplest form of hydromechanical concentration.

Efficiency of concentrating methods in the Joplin district.

Quantity of ore through mill, tons	309.70
Quantity of zinc concentrate produced, tons	9.235
Assay of zinc concentrate, per cent Zn	59.
Quantity of blende (ZnS, 67 per cent Zn) in concentrate, tons	8.132
Quantity of tailing, tons	300.465
Assay of tailing from mill (0.96 per cent Zn) per cent ZnS	1.43
Loss of blende (ZnS) in tailing, tons	4.296
Total quantity of blende (ZnS) in ore through mill, tons	12.428
Total recovery of blende through mill, per cent	65.43

The average loss, therefore, in the concentration of zinc in the Joplin district is 35 to 40 per cent. There are heavy losses in smelting. The total loss reaches nearly 50 per cent.

ELECTROMECHANICAL PROCESSES.

MAGNETIC SEPARATORS.

A magnetic separator contains a set of magnets by which paramagnetic or weakly magnetic minerals brought into the magnetic field are influenced in such a manner as to effect the desired separation. For example, a mixed sulphide ore crushed to the required degree of fineness and passed over a wet-concentrating table yields a galena-pyrite product and a pyrite-blende product. If the latter product be passed over a magnetic concentrator, such as the Wetherill, two products are obtained: (1) A blende product—assaying about 50 per cent zinc, 10 to 12 per cent iron, and 1 per cent lead—which is sold to the zinc smelters; and (2), a pyrite product, containing some lead and about 5 to 7 per cent zinc, which is mixed with the galena-pyrite product from the concentrating tables and is sold to the lead smelters, for whom it is a desirable ore. When this ore is smelted in the lead blast furnace, the zinc is lost in the slag and fume.

ELECTROSTATIC SEPARATORS.

Electrostatic separators are based on the difference in electrical conductivity of different minerals and on the attraction of bodies having opposite electrical charges. Most of the gangue minerals and some of the metallic sulphides, for example, blende, are poor conductors, whereas many metallic sulphides and other minerals are good conductors, as pointed out by Richards:[b]

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[a] Wright, C. A., Mining and treatment of lead and zinc ores in the Joplin district, Missouri, a preliminary report: Tech. Paper 41, Bureau of Mines, 1913, p. 36.

[b] Richards, Robert H., Ore dressing, vol. 3, 1909, p. 1549.

> If a mixture of good and poor conductors, in a neutral state, be dropped upon a highly charged conducting surface, the good conductors immediately receive a charge similar to that of the surface and are repelled, while the poor conductors are much more loath to receive the charge and therefore not so readily repelled. If, however, a material charged to a high potential of the opposite sign be fed to the above-mentioned surface, the good conductors, as before, assume immediately the condition of the charged surface and are repelled, while the very poor conductors, carrying a charge opposite to that which the surface carries, by the above law, tend to cling to that surface, thus making a sharper division of the separation. Theoretically the separation of two minerals does not require that one of them be a very good conductor and the other a very poor conductor, but merely that there be a difference in the degree to which they will conduct electricity. Commercially, of course, there are limits to this.

At Midvale, Utah, near Salt Lake City, at the plant of the United States Smelting, Refining & Mining Co., an electrostatic concentrating plant has been in operation since 1909, that is said to save 5,000-000 to 7,000,000 pounds of zinc annually which before had gone to waste. Zinc-iron middlings, partly furnished by the company's concentrator, partly from custom ore from the district, are passed over a Huff separator. MacGregor [a] states that the "feed" to the mill assays approximately 0.04 ounce gold, 2.5 ounces silver, 2.5 per cent lead, 1 per cent copper, 23 per cent iron, 28 per cent zinc; the separated zinc product assays 48 to 50 per cent zinc with 3 to 5 per cent iron, and the iron product 10 per cent zinc. Most of the gold, silver, copper, and lead is found in the iron product, and this product is smelted in the company's lead furnaces.

The iron product contained about 10 per cent zinc. The assays of other products, as given by MacGregor, show the same thing; that is, in the "iron product" or the "copper product"—the product that goes to the blast furnace and from which the greater part of the zinc has been separated—there is still anywhere from 8 to 12 per cent zinc, and the zinc in this product is not recovered.

FLOTATION PROCESSES.

In order to concentrate a zinc ore by flotation it is generally necessary to heat the pulp and to use acid. If the ore contains both lead and zinc the minerals carrying these metals are floated, and the lead-bearing minerals must be separated from the zinc-bearing minerals. In order to effect such a separation, Horwood, of Australia, devised a selective or preferential process. The following statement in regard to this process is taken from Metallurgical and Chemical Engineering [b].

> *Selective flotation by Horwood's process.*—The Broken Hill correspondent of the Australian Mining Standard, January 15, 1914, states that the successful establishment of

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[a] MacGregor, F. S., Progress in electrostatic ore dressing: Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 352.

[b] News item, Selective flotation by Horwood's process: Met. and Chem. Eng., vol. 12, May, 1914, p. 350.

the Horwood process of selective flotation at the works of the Zinc Corporation was one of the metallurgical achievements of 1913. The principle of this process is the roasting of mixed lead-zinc concentrates obtained by flotation, whereby the lead particles are slightly sulphatized while the zinc is unaffected. On floating the roasted product the zinc alone floats and the lead remains submerged.

A plant having a capacity of 500 tons weekly has been in operation for over six months, treating current and accumulated slime concentrates. The slimes are first dewatered and washed to remove soluble salts, then lightly roasted in an Edwards furnace, and finally treated again by flotation. Typical actual results from this plant are as follows:

	Zinc, per cent.	Lead, per cent.	Silver, oz. per ton.
Feed to roaster	40.4	14.6	21.4
Zinc concentrate	48.7	5.2	11.6
Lead residue	10.2	44.2	54.6

The grade of the lead is, of course, dependent on the amount of gangue in the feed. Tests made on Tasmanian ores show that 85 to 90 per cent of the zinc can be recovered as a concentrate assaying 57 to 58 per cent zinc, while the lead will remain with the iron content of the ore. From 75 to 80 per cent of the silver content of the ore and about 90 per cent of the gold will accompany the lead residue.

Other differential flotation processes that have been more recently developed are mentioned on a later page of this bulletin.

ZINC RECOVERY BY DIFFERENT CONCENTRATION PROCESSES.

Table 2 following shows the average recovery of zinc by various standard concentration processes. Most of the present processes rarely give a clean separation of lead and zinc; the product that goes to the zinc smelter contains considerable lead, and vice versa, the zinc contained in the lead product or the lead in the zinc product being lost. Only the zinc lost in smelting lead concentrates in a blast furnace is considered; the flotation of zinc ores is described in a later chapter.

Table 2.—*Recovery of zinc by various concentration processes.*[a]

Process.	Zinc in zinc concentrate, per cent.	Zinc in product going to lead-blast furnace, per cent.	Per cent recovered.			Remarks.
			Zn.	Pb.	Ag.	
Ordinary wet concentration:						
Daly West, Park City, Utah	....	....	50.0	95.0	75-80	
Daly Judge, Park City, Utah	....	....	66.0	92.0	78.0	
Buffalo Hump Mining Co., Burke, Idaho	....	....	25.0	90.0	85.0	
Joplin, Mo.	....	....	65.43	....	....	
Wetherill magnetic [b]	50.0	5-7	....	....	....	
Huff electrostatic [c]	48-50	10.0	....	....	....	
[c]	41.3	12.6	....	....	....	
	49.3	8.6	....	....	....	
Potter-Delprat [d]	....	....	66.2	38.5	50.9	
De Bavay [d] 1910	48.1	11.7	....	....	....	
1911	48.4	14.2	....	....	....	
1912	49.2	16.6	....	....	....	
Elmore vacuum [d]	46.0	15.84	....	....	....	
Minerals Separation process	....	....	85.4	74.8	80.1	
Horwood flotation process	48.7	10.2	....	....	....	
Hyde flotation process at Butte and Superior	....	....	88.1	....	83.4	For 1912.
			86.4	62.5	80.0	

[a] Lyon, D. A., and Arentz, S. S., Losses of zinc in mining, milling, and smelting: Trans. Am. Inst. Min. Eng., vol. 49, 1915, p. 799.

[b] Engineering and Mining Journal, Magnetic separation of zinc blende at Denver, Colo., vol. 74, Aug. 16, 1902, p. 217.

[c] MacGregor, F. S., Progress in electrostatic ore dressing: Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 352.

[d] Hoover, T. J., Concentrating ores by flotation, London, 1912, pp. 91, 97, 105.

LOSS OF ZINC IN LEAD SMELTING.

In smelting a zincky lead concentrate in a blast furnace the metallurgist seeks to get rid of the zinc either by fluxing it with the slag or by volatilizing it as fume, and this zinc is lost. Not only does the miner receive no return for any zinc in the lead concentrates he sends to the smelter, but he is penalized if the proportion of zinc is more than 8 to 10 per cent, because the presence of a larger percentage of zinc causes serious irregularities in the working of the lead furnace and decidedly increases the cost of smelting the ore, more coke being required per ton of ore smelted and the lead-producing capacity of the furnace being lowered. As Anderson[a] has pointed out, it is impossible to express in writing the difficulties met in attempting to smelt zincky ores.

To illustrate the amount of zinc lost in this country in smelting zincky lead ores, the following example[b] will serve:

A concentrate containing, say, 28 per cent of zinc is separated into two products, one containing, say, 50 per cent zinc and the other containing most of the gold, silver, copper, and lead, and about 10 per cent zinc. Then of the original 50 tons of concentrate there will be 28 tons (more or less, according to the percentage of iron present) of product containing 50 per cent zinc, and 22 tons containing 10 per cent, and in addition most of the gold, silver, copper, and lead. This product goes to the blast furnace, where ordinarily the zinc content is not saved but is lost in the fume and slag. If the daily tonnage treated is only 50 tons, the annual loss of zinc is almost 1,500,000 pounds. Moreover, this zinc is lost for all time, as it can not be recovered. Evidently the amount of zinc lost in this way aggregates thousands of tons annually. As Clevenger and others have indicated, the inability to recover this zinc constitutes one of the greatest weaknesses of present metallurgical practice, for the zinc so lost represents a distinct economic waste. In the prevention of this waste lies one of the sources from which in the future will be obtained a large part of our zinc supply. The same considerations, of course, hold true to some degree for lead, for any lead present in ores smelted in a copper blast furnace is volatilized and lost.

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[a] Anderson, L. D., Some methods of recovering zinc from its ores: Utah Soc. Eng., Salt Lake City, Utah, May, 1916.

[b] Lyon, D. A., and Arentz, S. S., Losses of zinc in mining, milling, and smelting: Trans. Am. Inst. Min. Eng., vol. 49, 1915, p. 798.

These facts are not to be considered as an indictment of electromechanical methods of concentration or of modern methods of copper and lead smelting; rather they are presented to emphasize the fact that although mechanical concentration and modern methods of smelting have reached a high stage of development, their use entails a great loss of zinc, and that in the future it will quite probably be necessary to stop these wastes in order to increase the available supply of zinc.

From the foregoing brief discussion of the subject the reader will note that certain zinc ores have not been treated for the recovery of the zinc because of the requirements of smelting in retorts, and these same requirements are indirectly responsible for the present losses in the mining of many zinc ores.

SCOPE OF INVESTIGATIONS AT SALT LAKE CITY STATION.

OUTLINE OF WORK.

As one of the functions of the Federal Bureau of Mines is to conduct investigations looking to the prevention of mineral waste, and as the research work being done by the Salt Lake City station of the Bureau of Mines, in cooperation with the department of metallurgical research of the State School of Mines of the University of Utah, is to gather and distribute such information as will make possible the treatment of low-grade and complex ores not now being treated, the problem of treating such zinc ores and the prevention of zinc waste was one of the first problems undertaken by the station.

In 1914 a preliminary survey was made of the State of Utah in order to ascertain the nature and extent of the various low-grade and complex ore bodies. This survey showed that large bodies of zinc ores in this State have not been treated because of their being too low grade or too complex. Large bodies of similar ores, especially oxidized and semioxidized ores, are in the adjoining States—Nevada, Idaho, and Colorado. Analyses of such ores are shown in Table 3. The complex zinc ores occur mostly as sulphides, and their composition is shown in Table 4.

TABLE 3.—*Analyses of oxidized and semioxidized zinc ores.*

Mine and sample.	District.	Fe.	CaO.	MgO.	Al_2O_3.	Zn.	Pb.	Cu.	Insol.	S.	CO_2.	Mn.	Ag.	Au.
		Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Ounces.*	*Ounces.*
Scranton, 1	Scranton, Utah	7.56	9.75	1.55	6.2	20.5	8.65	None.	12.9	0.2	15.90	0.04	0.45	Trace.
Scranton, 2	do	33.0	8.2	4.67	3.3	8.6	.77	None.	10.6	.35	13.19	.05	.28	Trace.
Scranton, 3	do	18.5	12.4	8.58	2.6	18.2	None.	.1	8.5	.5	23.57	.19	.16	Trace.
Scranton, 5	do	25.1	14.0	4.52		20.2	1.99		12.4					
Lower Mammoth, 1	Eureka, Utah	11.75	2.3	1.45	5.8	7.8	1.1	Trace.	52.7	6.5	2.12	.53	1.80	.03
Lower Mammoth, 2	do	4.79	.9	.40	4.5	12.0	.3	.4	59.6	1.6		.27	.3	None.
Lower Mammoth, 3	do	27.0	2.1	2.10	12.1	10.0	2.3	1.64	19.1	.4		.72	4.3	.03
May Day, 2	do	4.2	10.3	Trace.	None.	15.7	.6	None.	31.4	.7	19.84	None.	.3	.23
May Day, 4	do	2.6	12.2	8.65		20.7	10.9	None.	25.3	.2		1.07	2.78	.10
Bullion Coalition, 4	Stockton, Utah	15.0	1.1		8.6	29.9	.21	.13	12.5	.92	19.04		.15	Trace.
Bullion Coalition, 13	do	18.78	2.8	8.7	4.16	29.1	.30	Trace.	1.6	.41		.63		
Santaquin King	Santaquin, Utah	.33	None.	None.	Trace.	45.8	Trace.	Trace.	33.9					
Lake View, 104	Promontory, Utah	.3	16.1			21.9	12.0	None.	9.2	None.				
Lake View, 105	do	.8	19.5		1.53	31.3	1.4	None.	14.0	None.				
Lake View, 106	do	2.6	16.8		4.11	11.8	6.3	None.	7.6	None.				
Yellow Pine, 1	Goodsprings, Nev	1.4	4.6			32.7	14.6	Trace.	7.3				11.04	Trace.
Western Mining Co	Leadville, Colo	19.7	Trace.		3.3	20.1	.05	None.	19.5					
Trapper, tailing	Melrose, Mont	7.0	5.5	3.62	Trace.	11.3	2.58	.64	54.0	.30		.02	8.7	.055
Nevada Zinc Co	Tobar, Nev	5.3	17.35	10.75	.37	13.1	None.		13.3	.24		.15		
Yellow Pine, 2	Goodsprings, Nev	3.7	13.7	10.5	.32	18.5	4.70	.24	11.15	.21		.08	2.5	.02
Yellow Pine, 3	do	2.9	12.8	8.3	.37	19.2	3.12	None.	22.0	.25		.05	3.7	None.
Yellow Pine, 4	do	2.7	6.8	5.5	.28	29.3	9.32	None.	14.8	.31		.02	3.9	None.
Frederickson, 1	do	2.0	10.5	7.2	.19	12.2	5.85	None.	36.6	.30		.09	10.76	None.
Frederickson, 2	do	2.0	1.2	Trace.	.23	13.8	7.51	.03	64.5	.14		.005		None.
Monte Christo	do	2.0	6.3	3.9	.19	22.1	None.	None.	48.2	.21		.05	1.50	None.
Moscow, 4	Beaver County, Utah	7.5	5.1	3.8	3.41	20.6	.76	.49	29.8	.24			1.42	None.
Cedar Talisman, 2	do	3.4	2.3	2.7	1.56	25.7	2.85	.05	51.4	.14			2.36	Trace.

TABLE 4.—*Analyses of complex zinc-sulphide ores and concentrates.*

Mine and sample.	District.	Fe.	CaO.	MgO.	Al_2O_3.	Zn.	Pb.	Cu.	Insol.	S.	As.	CO_2.	Mn.	Ag.	Au.
		Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Ounces.*	*Ounces.*
Daly-Judge, 1	Park City, Utah	2.3	10.1	1.16	7.4	16.9	14.9	0.72	30.8	12.7	0.46	10.55	....	29.56	0.01
Daly-Judge, 2	do	16.4	5.1	Trace.	1.9	12.0	15.4	.07	22.0	24.8	.20	6.48	....	4.86	.04
Daly-Judge, 4	do	7.4	4.9	Trace.	5.2	46.0	1.86	1.54	4.1	30.4	.52	....	2.0	18.45	.015
Daly-Judge, 5	do	4.4	10.7	....	5.1	9.0	5.52	Trace.	50.5	8.5	....	....	....	....	....
Horn Silver, dump	Frisco, Utah	4.4	2.2	Trace.	8.5	6.3	7.4	.25	53.6	6.13	....	1.37	.87	6.70	.02
Ophir Hill Consolidated, 1 (tailing)	Ophir, Utah	3.4	4.2	....	....	2.2	Trace.	.47	67.5	2.78	....	....	....	1.64	....
Ophir Hill Consolidated, 2 (slimes)	do	4.3	5.0	....	....	3.9	3.83	1.16	51.5	4.69	.25	....	....	4.12	....
Bullion Coalition, 1	Stockton, Utah	33.0	2.1	Trace.	7.6	4.4	3.00	.10	10.5	37.34	1.13	3.52	....	.78	.01
Bullion Coalition, 2	do	28.8	3.9	....	15.8	8.5	10.47	.07	9.3	35.86	2.33	....	....	2.43	.02
Bullion Coalition, 3	do	28.8	2.1	Trace.	....	11.4	2.59	.06	14.0	33.26	1.69	3.53	....	1.06	.01
Bullion Coalition, 5	do	22.2	1.1	Trace.	14.6	13.1	10.56	.10	18.0	30.32	2.42	....	....	2.65	.02
Success	Deep Creek, Utah	16.4	1.4	Trace.	4.2	6.8	6.9	.60	38.2	19.70	....	....	....	40.59	.07
Bingham Mines Co	Bingham, Utah	8.6	....	....	....	4.53	6.85	.65	56.3	10.8	....	....	....	8.06	.01
United States smelter	Midvale, Utah	24.87	....	....	....	28.25	3.66	1.00	....	....	....	....	....	....	....
Mary Murphy, 1	Romley, Colo	2.8	....	....	....	39.5	11.7	2.12	14.6	27.2	....	....	.77	....	....
Mary Murphy, 2	do	11.0	....	....	....	31.8	11.6	1.14	7.5	37.6	....	....	1.38	....	....
Mary Murphy, 3	do	5.5	....	....	....	42.6	9.2	.57	7.5	31.9	....	....	2.29	....	....
Mary Murphy, 4	do	24.4	....	....	....	14.4	12.7	2.76	7.4	41.1	....	....	.91	....	....
Mary Murphy, 5	do	2.8	....	....	....	21.8	53.4	.86	.4	26.0	....	....	....	....	....
Success	Wallace, Idaho	....	....	....	....	....	....	....	....	....	....	....	....	....	....
Douglas	Pine Creek, Idaho	9.3	1.65	....	4.9	24.35	9.0	None.	30.4	....	....	....	....	....	....
Constitution, 1	do	7.4	5.47	....	11.3	32.3	12.5	None.	17.6	....	....	....	....	....	....
Highland Surprise	do	12.1	....	....	....	42.8	6.74	None.	4.9	22.0	....	....	....	....	....
Timber Butte (fines)	Butte, Mont	4.4	.75	....	3.6	53.9	1.53	2.82	3.6	....	.55	....	....	16.04	.04
Timber Butte (flotation)	do	8.6	1.88	....	8.6	48.6	1.85	1.85	3.6	....	.40	....	....	10.80	.03
North Star	Haley, Idaho	14.4	....	....	....	16.0	5.12	None.	....	18.62	....	....	....	6.6	....
Morning mill	Mullan, Idaho	8.3	1.79	....	7.3	44.5	10.40	....	33.4	16.4	....	....	....	....	....
Frisco mill, 1	Gem, Idaho	13.5	1.79	....	5.9	38.8	10.0	....	8.5	19.3	....	....	....	....	....
Frisco mill, 2	do	13.0	1.84	....	2.0	39.6	10.4	None.	2.0	26.3	....	....	....	....	....
Marsh mill, 1	Wallace, Idaho	....	....	....	....	....	....	....	....	....	....	....	....	....	....

The work reported in this bulletin is outlined by the table of contents (p. 3). It covers only the concentration of zinc ores and hydrometallurgy of zinc, the smelting of zinc ores not being considered.

The zinc ore investigations carried on thus far by the Salt Lake City station are as follows:

SULPHIDE ORES.

A. Mechanical concentration.
    1. Ordinary gravity concentration.
    2. Flotation processes.
        (*a*) Selective flotation.
        (*b*) Differential flotation.
B. Roasting for leaching or mechanical separation.
    1. Chloridizing roasting.
    2. Sulphate roasting.
        (*a*) In muffles.
        (*b*) In blast pots.
    3. Oxidizing roasting.
        (*a*) Present commercial work.
        (*b*) Work at University of Utah.
    4. Magnetic roasting.
C. Leaching processes.
    1. Acid solvents.
        (*a*) Sulphuric acid.
        (*b*) Sulphurous acid.
        (*c*) Hydrochloric acid.
        (*d*) Hydrofluosilicic acid.
    2. Alkaline solvents.
        (*a*) Caustic soda.
        (*b*) Ammoniacal solutions.
D. Precipitation processes.
    1. Electrolytic precipitation.
        (*a*) Development, and present status of process.
        (*b*) Application to various types of leaching solutions.
        (*c*) Zinc dust by electrolytic methods.
    2. Chemical precipitation.
        (*a*) From sulphate solutions.
        (*b*) From sulphurous acid solution.
        (*c*) From ammoniacal and alkaline solutions.
        (*d*) Miscellaneous methods.
E. Dry processes.
    1. Sulphate roasting and leaching.
    2. Igneous concentration.
    3. Magnetic separation.

OXIDIZED ORES.

A. Mechanical concentrating process.
    1. Ordinary milling processes, and why they are not applicable.
    2. Flotation processes.
        (*a*) Selective flotation.
        (*b*) Differential flotation.

- B. Roasting processes.
    - 1. Chloridizing roasting.
    - 2. Sulphate roasting.
- C. Leaching processes.
    - 1. Acid solvents.
        - (*a*) Sulphuric acid.
        - (*b*) Sulphurous acid.
        - (*c*) Hydrochloric acid.
        - (*d*) Hydrofluosilicic acid.
    - 2. Alkaline solvents.
        - (*a*) Caustic soda.
        - (*b*) Ammoniacal solutions.
- D. Precipitation processes.
    - (Same as for sulphide ores.)
- E. Dry processes.
    - Igneous concentration.

DISCUSSION OF THE WORK OUTLINED.

MECHANICAL CONCENTRATION OF ZINC SULPHIDE ORES.

Most of the zinc recovered from sulphide ores in Utah is from mixtures of lead and zinc sulphides. The usual milling treatment for such ores is to crush, size, or classify the ore, and then separate as cleanly as possible the gangue material from the sulphides with jigs and tables. By this separation of a mixed sulphide ore a lead product and a zinc product are obtained. Sometimes the lead product contains as much as 10 per cent zinc, or even more, as is shown by column 2 of Table 2 (p. 18), and the zinc product contains a considerable quantity of lead.

As previously stated, the zinc that accompanies any lead ore sent to the lead smelter is not recovered, and the problems connected with the milling of a zinc sulphide ore are as follows:

1. Effecting a better separation of the lead and zinc minerals.
2. Preventing losses of zinc, or of lead and zinc in the tailing.

The Bureau of Mines has studied this problem in Wisconsin[a] and in Missouri.[b] During the fiscal year 1915–1916 the department of metallurgical research of the Utah school of mines had one "fellow" giving his time to a study of the mining and milling of complex lead and zinc ores in the Park City district, Utah. The results of this investigation were embodied in a report[c] to the department, from which the following extract is taken:

> *Introduction.*—The Park City mining district is situated on the eastern slope of the Wasatch Mountains and 25 miles southeast of Salt Lake City, which is west of the mountains.
>
> This district has been a continuous and steady producer of lead-silver ores for 40 years.

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[a] Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin: Tech. Paper 95, Bureau of Mines, 1915, 39 pp.

[b] Wright, C. A., and Buehler, H. A., Mining and milling of lead and zinc ores in the Missouri-Kansas-Oklahoma lead and zinc district: Bull. 154, Bureau of Mines, 1918, 132 pp.

[c] Stott, G. F., Mining and milling of complex lead and zinc ores in the Park City district, Utah.

Production.—The total production of the district has been $156,500,000, of which $42,000,000 has been paid in dividends. In 1915 the district shipped 99,795 tons of crude ore and concentrates, valued at about $4,000,000.

Total production of metals up to January 1, 1915.[a]

Gold, ounces	4,058,315
Silver, ounces	125,707,056
Lead, pounds	1,136,145,128
Copper, pounds	25,943,492
Zinc, pounds	48,081,726
Total value	$152,454,585

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1. Broadwater mill, 2 miles below the town of Park City, 450,000 tons, averaging 1 per cent Pb, 4 per cent Zn, and 4–5 ounces Ag per ton.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	4.64
Silver, at 58 cents	2.61
Total	8.15

$8.15×450,000=$3,668,000, total value in minerals.

2. Beggs mill, one-half mile below the Broadwater mill: Deposit 2,500 feet long, 200 feet wide, and 4½ feet deep; 250,000 tons, assaying about the same as the Broadwater; total value of minerals, $2,018,000.

3. Big Four mill, 7 miles below Park City; deposit 3½ miles long, 500 feet wide, and 2 feet deep; total tonnage available estimated over 1,000,000 tons.

1.8 per cent Pb, 4.1 per cent Zn, 3.1 ounces Ag, Au 30 cents, 2½ per cent Fe, 62 per cent SiO_2.

	Per ton.
Lead, at 4½ cents	$1.61
Zinc, at 5.8 cents	4.75
Silver, at 58 cents	1.80
Gold	.30
Total	8.46

8.46×1,000,000=$8,460,000, total value in minerals.

Other small flats will bring the total up to over 2,000,000 tons of tailings worth $8 per ton, or $16,000,000, of which a recovery of about 60 per cent of the total mineral values may be expected.

TAILING LOSSES OF TO-DAY.

Even to-day with modern improvements and up-to-date milling practice the zinc losses are extremely large.

At the Daly West mill the "general tails" assayed for six months ending November 1, 1915, 1 per cent Pb, 3.05 per cent Zn, 2.04 ounces Ag, during which time an average of 146 tons per day were treated or approximately 21,900 tons.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	2.28
Silver, at 58 cents	1.18
Total	4.46

Economic loss, $433 per day or approximately $65,000 in a period of six months.

For the month of September, 1915, 1 per cent Pb, 2.36 per cent Zn, 2.28 ounces Ag, during which time 3,547 tons were treated or 154 tons per day.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	2.74
Silver, at 58 cents	1.32
Total	4.96

Loss, $482.5 per day or $11,700 total.

For the month of October, 1915, 1 per cent Pb, 3.1 per cent Zn, 2.67 ounces Ag. Tonnage milled was small, being only 1,381 or 125.6 tons per shift worked.

Owing to the shortage of water, the mill was operated every other day.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	3.60
Silver, at 58 cents	1.55
Total	6.05

Loss, $507 per day or $5,570 total.

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[illegible]

[illegible]

[illegible]

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The Bureau of Mines has studied this problem in Wisconsin[a] and in Missouri.[b] During the fiscal year 1915–1916 the department of metallurgical research of the Utah school of mines had one "fellow" giving his time to a study of the mining and milling of [illegible] and zinc ores in the Park City district, Utah. The [illegible] investigation were embodied in a report[c] to the [illegible] which the following extract is taken:

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Gold, ounces	4,058,315
Silver, ounces	125,707,056
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Copper, pounds	25,943,492
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Silver, at 58 cents	2.61
Total	8.15

$8.15×450,000=$3,668,000, total value in minerals.

2. Beggs mill, one-half mile below the Broadwater mill: Deposit 2,500 feet long, 200 feet wide, and 4½ feet deep; 250,000 tons, assaying about the same as the Broadwater; total value of minerals, $2,018,000.

3. Big Four mill, 7 miles below Park City; deposit 3½ miles long, 500 feet wide, and 2 feet deep; total tonnage available estimated over 1,000,000 tons.

1.8 per cent Pb, 4.1 per cent Zn, 3.1 ounces Ag, Au 30 cents, 2½ per cent Fe, 62 per cent SiO_2.

	Per ton.
Lead, at 4½ cents	$1.61
Zinc, at 5.8 cents	4.75
Silver, at 58 cents	1.80
Gold	.30
Total	8.46

8.46×1,000,000=$8,460,000, total value in minerals.

Other small flats will bring the total up to over 2,000,000 tons of tailings worth $8 per ton, or $16,000,000, of which a recovery of about 60 per cent of the total mineral values may be expected.

TAILING LOSSES OF TO-DAY.

Even to-day with modern improvements and up-to-date milling practice the zinc losses are extremely large.

At the Daly West mill the "general tails" assayed for six months ending November 1, 1915, 1 per cent Pb, 3.05 per cent Zn, 2.04 ounces Ag, during which time an average of 146 tons per day were treated or approximately 21,900 tons.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	2.28
Silver, at 58 cents	1.18
Total	4.46

Economic loss, $433 per day or approximately $65,000 in a period of six months.

For the month of September, 1915, 1 per cent Pb, 2.36 per cent Zn, 2.28 ounces Ag, during which time 3,547 tons were treated or 154 tons per day.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	2.74
Silver, at 58 cents	1.32
Total	4.96

Loss, $482.5 per day or $11,700 total.

For the month of October, 1915, 1 per cent Pb, 3.1 per cent Zn, 2.67 ounces Ag. Tonnage milled was small, being only 1,381 or 125.6 tons per shift worked.

Owing to the shortage of water, the mill was operated every other day.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	3.60
Silver, at 58 cents	1.55
Total	6.05

Loss, $507 per day or $5,570 total.

Sample taken by Stott, November 3, 1915, 0.88 per cent Pb, 2.46 per cent Zn, 1.8 ounces Ag, 4.33 per cent Fe, 79.9 per cent insoluble:

	Per ton.
Lead, at 4½ cents	$0.79
Zinc, at 5.8 cents	2.85
Silver, at 58 cents	1.04
Total	4.68

At the Daly Judge mill the mill tailing assayed, before the installation of the flotation process, 1 per cent Pb, 3 per cent Zn, and 2.6 ounces Ag. The average tonnage for the year 1914 was 118 tons per day and a total of 49,095 dry tons milled.

	Per ton.
Lead, at 4½ cents	$0.90
Zinc, at 5.8 cents	3.48
Silver, at 58 cents	1.51
Total	5.89

Loss per day (78 tons)=$459.
Loss per year=$191,300.

At the Silver King Coalition the mill tailings assay as follows: General mill tailing from sulphide ores, 0.7 per cent Pb (fire assay), 2.3 ounces Ag, 2.6 per cent Fe (Jan. 10–12, 1916).

	Per ton.
Lead, at 4½ cents	$0.63
Silver, at 58 cents	1.33
Total	1.96

General mill tailing from oxide ores, 3.64 per cent Pb., 4.5 ounces Ag.

	Per ton.
Lead, at 4½ cents	$2.36
Silver, at 58 cents	2.61
Total	4.97

CREEK LEASING.

The above figures represent the material that is daily being sluiced into the creek as tailing. As a result we find to-day two small leasing mills immediately below each of these larger mills. There have been as many as 10 plants operating on the creek in series but they have been forced to quit on account of the mills above improving their extraction within the last year or two. The two surviving mills have been fairly well designed and are better equipped to handle the poorer grade of tailing.

A good insight into the past milling practice was obtained from Mr. Charles Moore, the owner of the two surviving "Creek mills."

In 1900 leasers placed sluice boxes with riffles in the creek. A fair shipping concentrate was obtained. The next step consisted of plank riffles across the creek. Crude concentrates were shoveled into a wagon and hauled down to a rather crude mill consisting of one Wilfley table. With one man and a team and another man operating the Wilfley table the leasers were able to make as high as $50 a day and averaged $30 apiece. Thus we can get some idea of how we can to-day go 7 miles below town and find 2,000,000 tons of tailing deposits worth $8 per ton.

Up to the last three or four years the tailing from the creek assayed 3 per cent Pb, 7 per cent Zn, and about 10 ounces Ag, indicating that the Daly West and Daly Judge mills were not extracting more than 50 to 60 per cent of the lead, 20 to 30 per cent of the zinc, and 35 to 40 per cent of the silver unless the heading to the mills was considerably higher in metallic values than it is at the present time.

MOORE CREEK LEASING MILL.

This mill built by Mr. Charles Moore at a cost of about $8,000 is a result of 14 years' experience at "creek leasings."

One special feature to be noticed in this mill is the manner of classification of the sands and the entire absence of crushing machinery.

Total average heading to this mill is about 275 tons of solids per day, assaying 1 per cent Pb, 2 ounces Ag, and 2.5 to 3 per cent Zn. Fifty per cent of this amount is removed as oversize at the first trommel and first screen; 15 per cent of the total is slime overflow and is lost on the water wheel.

Water in the creek averages about 1,500 gallons per minute for the year. During the early spring it is impossible to settle any slime on account of heavy flow of water from the melting snows. This lasts for four or five weeks. From the first of March until the first of August there is enough water in the creek to furnish power for running the mill. At other times it is necessary to run a 7-horsepower motor in addition to the water wheel, both of which are belted to a common line shaft, which drives all the machinery in the mill.

A small dam, 2 feet high, catches the creek water and diverts it into a 12-inch by 13-inch launder. At the mill there is a one-half-inch screen set in the launder at about 45° to the direction of the flow, and this removes all one-half-inch oversize without allowing it to settle in the tanks.

Large settling tanks check the velocity of the water and act as classifiers. The first settling tank is 19 feet by 19 feet by 7 feet high. Nearly all of the material which is coarser than one-half inch is caught in the first two compartments of this settling tank. The first two compartments discharge into the trommel screens, and the second two compartments discharge into a box with a one-half-inch screen to catch any oversize and wood chips that might be carried over. From the box the undersize goes into the second settling tank, 10 feet by 12 feet by 6½ feet high, which has three compartments and discharges.

Middlings from each table are shoveled into a hopper and are flushed back to the head of the table for re-treatment. Lead-iron concentrates and the zinc concentrates go into separate boxes at the end of the Wilfley tables, and from there they are shoveled into a wheelbarrow and then to their respective bins. When enough concentrate is stored up to make a shipment it is shoveled into a wagon and hauled to the railroad.

The average output of this mill in 24 hours is 1 ton of zinc concentrates and about one-half ton of lead-iron concentrates. During the year 1914 this mill shipped 400 tons of zinc concentrates and 300 tons of lead-iron concentrates.

Smelter contract calls for:

Payments.

Gold, $20 for 0.03 or over.
Silver, 95 per cent of the total if over 1 ounce.
Lead, 60 per cent of the lead.
Copper, less 0.75 per cent Cu less 4½ cents per pound.
Iron, 10 cents per unit.

Charges.

Insoluble, 10 cents per unit.
Sulphur, 5 per cent allowed free, excess 25 cents per unit, $3 maximum.
Zinc, 5 per cent allowed free, excess 20 cents per unit, $3 maximum.
Speiss, 5 per cent allowed free, excess 25 cents per unit.
Base treatment charge: $2 on a basis of 30 per cent Pb.
Paid 10 cents a unit for all over 30 per cent Pb.
Charges 10 cents a unit for all under 30 per cent Pb.

Thus, for the lead-iron concentrate assaying 20 per cent Pb, 15 per cent Fe, 1.3 per cent Cu, 15 per cent Zn, 25 ounces Ag, 5 per cent SiO_2, and 25 per cent S, the following would be the smelter returns (normal prices):

Payment.		Charges.	
Lead	$10.80	Sulphur	$2.50
Iron	1.00	Zinc	2.00
Copper	1.10	Freight	2.00
Silver	13.78	Sampling	.50
		Hauling to railroad	.50
Total	26.68		7.50
Charges	7.50		
Net from sale of 1 ton of concentrates	19.18		

Smelter contract with the Iola, Kans., zinc smelter up until the present year paid:

Payment.

Zinc, 80 per cent of zinc on a basis of 35 per cent Zn or better.

Silver, 90 per cent of the silver if 1 ounce or over.

Charges.

Treatment charge, $15 per ton for 35 per cent Zn or better.

Freight to smelter, $6.80 per ton to Iola, Kans.

Thus for the zinc concentrate assaying 2 to 3 per cent Pb, 8 per cent Fe, 39 per cent Zn, 0.5 per cent Cu, 13 ounces Ag, and 20–25 per cent S, the following would be the smelter returns, assuming normal prices:

Payment.		Charges.	
Zinc	$36.20	Smeltering	$15.00
Silver	6.79	Freight	6.80
		Sampling	.50
	42.99	Hauling to railroad	.50
	22.80		22.80
Net from sale of 1 ton of concentrate	20.19		

Costs per day.

Power, $2 per day for 7½-horsepower motor	$2.00
Rent of ground on creek, $50 per month	1.66
Three men operating mill, at $3.50 per day	10.50
Repairs, heating, and extras	.50
Total cost per day	14.66

Average output of the mill is 1 ton of zinc concentrate and about one-half ton of lead-iron concentrate. From the sale of these concentrates—

Lead-iron concentrate (½ ton)	$9.59
Zinc concentrate (1 ton)	20.19
Total	29.78
Total cost per day	14.66
Profit	15.12

on an investment of $8,000. If the mill works 300 days a year this represents $4,536, or 56.6 per cent of the money invested.

DALY WEST MILL.

Ore from the mine is divided into two classes, milling and crude ore, according to the amount and character of the minerals contained. Zinc and silver are the determining factors. For instance, an ore containing over 15 per cent Pb and 20 ounces of Ag is usually shipped as a crude ore; however, if it contains a high percentage of zinc this ore may be sent to the mill in order to separate the zinc from the lead minerals.

A smelter contract with the American Smelting & Refining Co. smelter at Murray calls for a very high base treatment charge and does not penalize for zinc. Therefore this mining company is not very careful to save the lead-zinc ores, especially if they contain high silver values.

Both the crude shipping and the milling ores are complex ores of lead, iron, silver, zinc, and some copper in a gangue of quartz. As the oxidized zone does not extend below the 400-foot level, all of the ores now being shipped are sulphides.

The lead occurs as coarse and granular galena; zinc occurs in various forms, the usual variety is the yellow resinous type of sphalerite; another type is the granular sphalerite (yellow), also resinous but very friable. Pyrite here found exists as small crystals in relatively small amounts. Copper is present only in very small amounts, but is found both as tetrahedrite and chalcopyrite. The silver is found existing in chemical and mechanical combinations with all of the above minerals.

The ore is milled in a plant containing jigs, tables, and flotation machines, the latter being a more recent addition to the plant. Some data on this mill which is of interest is the following:

For the month of May, 1915, the costs for the flotation plant:

Pay roll, $150; power, $81.60; supplies, $15.98. This makes a total cost of $247.58. Assuming that the plant treated 25 tons per day of 31 days, the cost per ton of dry slime is $0.313.

For the period of eight months ending November 1, 1915:

	Pb, per cent.	Ag, ounces.
Flotation heading	6.36	11.45
Tailing	.59	2.84
Concentrate	25.80	29.75
Extraction	92.45	[a] 81.00
For the month of September, 1915:		
Flotation heading	5.65	11.28
Tailing	.44	2.88
Concentrate	29.95	42.64
Extraction	93.7	[a] 79.9
For the month of October, 1915:		
Flotation heading	9.15	14.84
Tailing	.35	2.72
Concentrate	24.80	36.52
Extraction	97.5	[a] 88.0

Test of the flotation efficiency by Stott, Nov. 3, 1915.

	Pb	Zn	Fe	SiO_2	Ag	Cu
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Oz.*	*Per cent.*
Heading	8.2	7.25	5.27	68.64	11.3	0.545
Tailing	.55	1.92	2.91	79.64	0.9	0.136
Concentrate	15.40	16.26	8.39	24.15	39.0	2.180
Extraction	96.5	83.4	....	20.4	[a] 94.3	80.6

[a] Per cent.

Test made on the Frantz jig efficiency by Stott, Nov. 3, 1916.

	Pb	Zn	Fe	Insoluble	Ag
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Oz.*
Heading	3.5	4.4	7.1	71.3	5.1
Concentrate	50.6	10.1	6.54	8.36	44.35
Tailing	0.83	2.46	5.05	78.92	1.0
Extraction	78.1	56.6	36.8	12.7	[a] 82.6

[a] Per cent.

Product to the mill is divided among the following ratios:

	Pb, per cent	Zn, per cent.	Ag, per cent.
Extraction, total of jigs and tables combined	84.0	50.0	62.5
Extraction increased by flotation	12.	10.8	10.5
Total metallic extraction	95.9	60.8	72.5

There is not enough data at hand to calculate the losses after concentration, but from all appearances the results will be rather startling. For instance, an analysis of the concentrates shows:

	Pb, per cent.	Zn, per cent.	Fe, per cent.	Insol., per cent.	Ag, ounces.
Lead-iron concentrate	36.42	7.4	13.29	14.15	38.2
Zinc concentrate	3.32	37.6	7.65	26.8	33.7

The loss of zinc in the lead concentrate and of lead in the zinc concentrate is a very common loss.

DALY-JUDGE MILLING PRACTICE.

Ore from the mine is divided into two classes, milling and crude ore, according to the amount and character of the minerals contained. Zinc and silver are the determining factors. For instance, an ore containing 12 to 15 per cent lead and less than 10 per cent zinc and 20 ounces of silver is shipped as crude lead ore. If it contains less lead or a higher percentage of zinc it is more profitable to treat in the mill and separate the lead and zinc minerals as far as possible.

Crude and milling ores are complex lead, zinc, iron, copper, and silver ores. These sulphide ores are identical with those found in the Daly-West. Oxidation is limited to the 500-foot level except in fissure deposits where the copper sulphides especially have been altered to carbonate and oxides, yielding azurite, cuprite, and little chrysocolla. The depth of oxidation in the fissures is limited to the 1,200-foot level. Below this point oxidation is not important.

The mill was designed to treat 150 tons of dry ore in eight hours. At the present time the mill is operating two shifts of eight hours each and is treating nearly 200 tons per day.

Milling costs, per ton.

	1914 (49,096 tons treated).	1915 (63,935 tons treated).
Labor	$0.386	$0.3935
Repairs and maintenance	.088	.2230
Supplies	.1083	.2063
Assay office	.0366	.0292
Fuel and electric power	.1150	.1140
Sundries	.0381	.0326
Cost per ton	.7720	.9986

SLIME TREATMENT AT THE DALY-JUDGE.

Slime treatment in the past has been rather difficult owing to lack of suitable machinery. All the slime prior to November 1, 1915, was treated on Wilfley fine sand tables and Wilfley double deck "slimers."

Average table concentration of slime feed for six months ending June 1, 1915:

TABLE 6.—*Mill tests at the Daly-Judge mill.*

[Previous to oil flotation.]

Material.	Assays.					Total contents.			
	Weights.	Pb	Zn	Ag	Cu	Pb	Zn	Ag	Cu
		Per cent.	*Per cent.*	*Oz. p. ton.*	*Per cent*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Jig and Pb concentrates	14.27	44.80	10.4	40.1	1.45	68.1	14.0	40.9	27.2
Jig and Zn concentrates	2.32	6.6	45.0	30.5	2.03	1.6	9.86	5.1	6.2
Jig tailing to tables	71.10	3.8	10.9	10.1	.66	23.6	73.1	51.1	63.4
Jig tailing to tailrace	12.31	1.2	2.6	3.4	.20	1.6	3.0	3.0	3.2
Heading	100.00	9.4	10.6	14.0	.76	......	100	......	......
All table Zn concentrate	11.66	2.3	45.0	24.2	1.82	2.8	49.6	20.7	27.9
Pb concentrate, sand and slime tables	7.41	22.5	13.0	35.1	2.03	17.7	9.1	18.6	19.8
Pb concentrate, upper slime tables	.41	20.5	21.7	37.1	2.25	.9	.8	1.1	1.2
Pb concentrate, basement slime tables	.34	26.8	13.0	35.9	2.03	1.0	.4	.9	.9
Pb concentrate, Wilfley slime	.09	30.0	6.9	28.2	1.34	.3	.1	.2	.2
Table tailing	51.19	1.1	2.7	2.8	0.20	5.9	13.2	10.2	13.5

Recovery, based on content of ore milled.

Silver (Ag)	per cent..	81
Gold (Au)	do....	....
Lead (Pb)	do....	84.6
Copper (Cu)	do....	76.1
Zinc (Zn)	do....	55.4

Recovery in lead and zinc concentrates.

		In lead concentrates.	In zinc concentrates.
Total Pb recovered	per cent..	56.2	1.25
Total Zn recovered	do....	9.7	30.50
Total Ag recovered	do....	35.0	10.80
Total Cu recovered	do....	27.2	15.70

FLOTATION.

Owing to the complex nature of the minerals in the slime feed it has been found impossible to make a differential flotation, but it has been found necessary to float all the sulphides possible and then resort to separation on Wilfley tables. Thus we find a very complicated flow sheet, although flotation ordinarily tends to simplify operations.

A 3-deck Dorr thickener, 30 feet in diameter, takes the slime overflow from the settling tank and three Richards-Janney classifiers. The thickened product contains approximately 15 per cent solids, of which 90 per cent will pass an 80-mesh screen. This seemingly thin pulp meets the flotation requirements to best advantage.

A centrifugal pump elevates the pulp from the thickener to a mixing box, where the oil is added and a second centrifugal pump discharges the pulp into a 20-foot Pachuca tank, 4 feet in diameter, giving further mixing before going to the Callow flotation cells.

The rate of oil feed is regulated by a small plunger pump whose speed can be altered by means of variable-speed pulleys. The length of stroke can also be varied by manipulating a lock nut, thus giving a good control of the amount of oil fed to the pulp as it is discharged from the first centrifugal pump.

After thorough mixing in the Pachuca tank the pulp is distributed to four roughing cells. The froth is collected and cleaned in the first cleaner cell. Tailing from this cell is re-treated in another cell before going to the tailrace.

Tailing from the roughing cells is thickened in Callow cones, and receives further treatment on two Wilfley tables. The product caught on these tables is the coarser material which is not floated. Each table yields lead concentrate, zinc middling, and tailing. The zinc middling is re-treated on a separate Wilfley table.

Considerable difficulty is found in breaking up the flotation froth without using too much water. A patent nozzle known as the Koerting spray has been found very effective in "killing" froth. The froth after passing under the spray is thickened in two Callow cones and the product fed to two Wilfley tables. Each table yields lead-iron concentrate and zinc middling. The zinc middling goes to a centrifugal pump and is elevated to a third Wilfley table. This table yields zinc concentrate and middling, which is returned to the head of the table for re-treatment.

This manner of slime treatment yields concentrate having the following analysis:

Results of analysis of slime concentrate.

	SiO_2.	Zn.	Pb.	Fe.	Ag.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Ounces.*
Slime heading to flotation	....	4.0	2.0	4.0	4.0
Lead-iron concentrate	1.5	7.0	34.0	16.0	36.0
Zinc concentrate	3.5	46.0	4.5	4.0	30.0
Apparent extraction	....	59.8	76.0	....	[a] 53.3

[a] Per cent.

Zinc concentrate is settled in two settling boxes and the product sent to the zinc concentrate bins.

Lead concentrate is thickened in an 8-foot Callow cone before being de-watered in a small settling box, and later taken to the lead-concentrate bins.

SILVER KING COALITION MILLING PRACTICE.

The ore is divided into two classes: First class, crude shipping ore, which contains values in lead and silver to exceed $17 per ton; second, milling ore which is less than $17 in value.

The first-class ore is a combination of lead, copper, and iron, with a high silver value, the silver being associated with pyrite and galena. There is more or less sphalerite and copper sulphides associated with the pyrite and the galena, but not in the same amount and extent as is found in Daly-West and Daly-Judge mines.

Ore bodies have been oxidized almost completely to the 900-foot level, and oxidation extends even to the 1,200-foot level. The case is rare when oxides are found below this point. Thus the milling ore is divided into two classes—oxide and sulphide ores in the ratio of two-thirds sulphide ores and one-third oxidized or "carbonate" ores, as they are termed locally.

In general the practice is to treat these ores separately. The mill bins hold large tonnages of ores, thus the mill will run for several days on sulphide ores and then will operate on carbonate ores. Each received the same treatment in crushing and jigging. Ores are crushed to ½ inch and treated on Hartz jigs. The jig tailing is reground in Marcy ball mills to pass a 30-mesh Callow screen. From this point on the treatment will depend on whether the ore is sulphide or carbonate. Sulphide ores are sent to a Callow system of flotation and carbonate ores are classified and concentrated on Wilfley and Deister tables.

SULPHIDE SECTION.

Undersize from the 30-mesh Callow screens goes to three flotation units, all alike, and each will be described as one individual unit.

Oil is added, a drop at a time, as the pulp is discharged from the Callow screen. One thing to be noted here is the absence of any pulp thickeners. The water ratio in the pulp is controlled by the water fed to the Marcy mills. The pulp dilution is carried at about 20 per cent dry solids, as nearly as can be approximated.

A box distributor sends the pulp to three 3 by 14 foot Pachuca tanks, each of which feeds a flotation unit. In the Pachuca tanks the oil and pulp are emulsified. Two rougher cells receive the pulp from the Pachuca tank. The rougher tailing is elevated by two 4-inch air lifts to a second series of two rougher cells. These cells make froth concentrate which received further re-treatment in one recleaner cell. Tailing from the second rougher is sent to waste. Tailing from the recleaner cell is sent to a small two-spigot V-classifier and two tables. The products from this table are concentrate and tailing which is sent to waste. Concentrates from the first rougher cell, from the recleaner cell, and from the two middling tables are combined and elevated by a centrifugal pump to two 8-foot Callow cones. The thickened product is filtered by an 8 by 8 foot Portland filter.

Extractions previous to the installation of the Callow flotation process, when using Wilfley tables and vanners, were—lead, 96 per cent; silver, 82 per cent.

After the installation of flotation the total mill extraction dropped to—lead, 88 per cent; silver, 73 per cent.

This difference is due largely to the attempt at floating a 30-mesh product. Lead sulphide does not float when it is as coarse as this.

Oil used is "1580 Pensocola," and approximately 1.15 pounds is used per ton of dry pulp.

These extractions are taken from assay sheets for only two or three days and do not represent any long period of time. The ore to the mill is very changeable and these extractions may be misleading, but will serve in one way to show what this mill is doing. The same applies to the extractions and data on the carbonate-ore treatment.

Data on treatment of sulphide ores.

Kind of material.	Pb	Ag	Fe	SiO_2
	Per cent.	*Ounces.*	*Per cent.*	*Per cent.*
Heading to flotation	2.4	5.7	5.6	....
Concentrate to flotation	24.2	40.0	13.8	....
Tailing to flotation	0.5	1.8	3.6	....
Extraction	80.8	[a] 71.6	40.4	....
Heading to jigs	6.4	10.8	10.4	....
Concentrate to jigs	20.6	24.4	29.5	....
Tailing to jigs	2.4	5.7	5.6	....
Extraction	61.6	[a] 76.8	57.8	....
General mill heading	6.2	5.3	8.8	54.2
General mill concentrate	22.6	34.2	14.3	....
General mill tailing	.7	2.3	2.6	....
SILVER KING MILL ORE. [Jan. 11, 12, and 13, 1916.]				
General mill heading	5.0	6.46	....	76.9
General mill concentrate	35.5	26.08	....	....
General mill tailing	1.0	3.3	....	....
Extraction	82.4	[a] 54.5	....	....

[a] Per cent.

Slime treatment of carbonate ores.

	Pb, per cent.	Ag, ounces.
Deister heading	3.7	5.9
Deister concentrate	37.0	38.3
Deister tailing	2.64	4.5
Extraction	32.0	[a] 26.9
Ratio of concentration for carbonate ores		8 ::1
Ratio of concentration for sulphide ores		4½ ::1

[a] Per cent.

COSTS.

The last annual report published by this company was in 1913. From this report we find that for the year ending April 30, 1913, 81,996 tons of second class or milling ore were treated with the production of 12,996 tons of concentrate assaying: 35.55 per cent Pb, 55.76 ounces Ag, 0.0648 ounce Au.

Ratio of concentration, 6.5 into one.

Mill treated 230 tons per day.

Costs per ton of ore treated.

Labor	$0.317
Supplies	.011
Pumping	.046
Teaming	.004
Power	.083
Heating and light	.018
Repairs	.335
Assaying	.022
Superintendency	.088
Insurance	.008
Tailing release	.014
	.868

MILLS ON TAILING FLATS BELOW PARK CITY.

Below the town of Park City there are two large and two smaller mills working over the tailings that have been discarded from Park City mills since 1886. The extent and value of these flats has been discussed under "Early Milling" and "Tailing Leases."

BROADWATER MILL.

This mill is treating about 400 tons of tailing per day, but has a capacity of about 500 tons per 24 hours. From 900 to 1,200 tons per day are taken to the mill. One-half to two-thirds of this tonnage is stored as a "winter dump" in order that the mill may operate during the winter months when the ground is frozen.

Heading to the mill assays: Pb, 1 per cent; Zn, 4 per cent; and Ag, 4.5 ounces.

Lead-iron concentrates assay: Pb, 20 per cent; Zn, 12 per cent; Ag, 20 ounces; Fe, 20 per cent; SiO_2, 4 per cent.

Zinc concentrates assay: Pb, 6 per cent; Zn, 37 per cent; Fe, 11 per cent; SiO_2, 8 per cent.

Zinc concentrate is being stored for treatment in the Daly-Judge electrolytic zinc plant which will probably be in operation by the time this report appears in print.

BEGGS MILL.

This mill is located about one-half mile below the Broadwater mill.

These tailing deposits were located by Mr. Beggs in 1907 as placer claims. Six claims in all were located on this flat. The deposit varies from 3 to 6 feet deep, and is 200 feet wide and 2,500 feet long.

Tailing from the flat is scraped up into heaps by means of drag-line scraper operated by a steam engine and requiring a crew of two men. The tailing is then shoveled by hand into 16-cubic foot mine cars and trammed to the hopper at the mill.

Tailings from the table were being discharged in so many different and inaccessible places that it was found impossible for us to obtain even a representative tailing sample, so none was taken. However, from data obtained from Mr. Beggs the extraction has been calculated:

	Pb.	Zn.	Ag.
Total extraction..........per cent..	63.5	61.1	30.2
Total received..........do....	40.75	50.6	30.2

Mill treats approximately 50 tons per day.

Average heading to mill—Pb, 1.5 per cent; Zn, 4 per cent; Ag, 3.5 ounces.

Average [illegible] concentrates [illegible]
[illegible] tonnes [illegible] per [illegible]
Average [illegible] concentrates [illegible]
[illegible] tonnes [illegible] per [illegible]

Mill [illegible] concentrates, [illegible] concentrates per [illegible]

[illegible]		
Labor	[illegible]	[illegible]
Power	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
Total [illegible]	[illegible]	[illegible]
Total [illegible]	[illegible]	[illegible]

Calculating [illegible] (assuming [illegible]).

	[illegible]		[illegible]
Gold	[illegible]	[illegible]	[illegible]
Silver	[illegible]	[illegible]	[illegible]
Lead	[illegible]	[illegible]	[illegible]
Iron	[illegible]	[illegible]	[illegible]
Total	[illegible]	[illegible]	[illegible]
Charges	[illegible]	[illegible]	[illegible]
Net from sale of [illegible] concentrates	[illegible]	Total	[illegible]

	Zinc concentrates		[illegible]
Gold	[illegible]	[illegible]	[illegible]
Silver	[illegible]	[illegible]	[illegible]
Zinc	[illegible]	[illegible]	[illegible]
Total	[illegible]	[illegible]	[illegible]
Charges	[illegible]	Total	[illegible]
Net from sale of [illegible] of zinc concentrate	[illegible]		

Then from above:

50 tons of [illegible] per day [illegible]	[illegible]
[illegible] tons of Pb [illegible] concentrates [illegible]	[illegible]
2.[illegible] tons of zinc concentrates [illegible]	[illegible]
Profit per day	[illegible]

BIG FOUR EXPLORATION CO. MILL

This mill is located at Atkinson, [illegible]

The portion of tailing deposits [illegible] an area [illegible] miles long, [illegible] feet wide, and averaging [illegible] able is in excess of [illegible] tons. [illegible] indicate that the average metal content of [illegible] Zn [illegible]

The tailing is [illegible] side of the railroad. From these [illegible] the tracks and [illegible] ton cars. [illegible] which acts as the [illegible]

The [illegible] was operated [illegible] a few [illegible] during which time it has been remodeled and changed. [illegible]

COSTS.

The last annual report published by this company was in 1913. From this report we find that for the year ending April 30, 1913, 81,996 tons of second class or milling ore were treated with the production of 12,996 tons of concentrate assaying: 35.55 per cent Pb, 55.76 ounces Ag, 0.0648 ounce Au.

Ratio of concentration, 6.5 into one.

Mill treated 230 tons per day.

Costs per ton of ore treated.

Labor	$0.317
Supplies	.011
Pumping	.046
Teaming	.004
Power	.083
Heating and light	.018
Repairs	.335
Assaying	.022
Superintendency	.088
Insurance	.008
Tailing release	.014
	.868

MILLS ON TAILING FLATS BELOW PARK CITY.

Below the town of Park City there are two large and two smaller mills working over the tailings that have been discarded from Park City mills since 1886. The extent and value of these flats has been discussed under "Early Milling" and "Tailing Leases."

BROADWATER MILL.

This mill is treating about 400 tons of tailing per day, but has a capacity of about 500 tons per 24 hours. From 900 to 1,200 tons per day are taken to the mill. One-half to two-thirds of this tonnage is stored as a "winter dump" in order that the mill may operate during the winter months when the ground is frozen.

Heading to the mill assays: Pb, 1 per cent; Zn, 4 per cent; and Ag, 4.5 ounces.

Lead-iron concentrates assay: Pb, 20 per cent; Zn, 12 per cent; Ag, 20 ounces; Fe, 20 per cent; SiO_2, 4 per cent.

Zinc concentrates assay: Pb, 6 per cent; Zn, 37 per cent; Fe, 11 per cent; SiO_2, 8 per cent.

Zinc concentrate is being stored for treatment in the Daly-Judge electrolytic zinc plant which will probably be in operation by the time this report appears in print.

BEGGS MILL.

This mill is located about one-half mile below the Broadwater mill.

These tailing deposits were located by Mr. Beggs in 1907 as placer claims. Six claims in all were located on this flat. The deposit varies from 3 to 6 feet deep, and is 200 feet wide and 2,500 feet long.

Tailing from the flat is scraped up into heaps by means of drag-line scraper operated by a steam engine and requiring a crew of two men. The tailing is then shoveled by hand into 16-cubic foot mine cars and trammed to the hopper at the mill.

Tailings from the table were being discharged in so many different and inaccessible places that it was found impossible for us to obtain even a representative tailing sample, so none was taken. However, from data obtained from Mr. Beggs the extraction has been calculated:

		Pb.	Zn.	Ag.
Total extraction	per cent..	63.5	61.1	30.2
Total received	do....	40.75	50.6	30.2

Mill treats approximately 50 tons per day.

Average heading to mill—Pb, 1.5 per cent; Zn, 4 per cent; Ag, 3.5 ounces.

Average lead-iron concentrate: Pb, 23 per cent; Zn, 16 per cent; Ag, 20 ounces; Au, 0.05 ounce; Fe, 18.5 per cent; SiO_2, 3 per cent.

Average zinc concentrate: Pb, 6.4 per cent; Zn, 38 per cent; Ag, 10 ounces; Au, 0.03 ounce; Fe, 9.5 per cent; SiO_2, 10 per cent.

Mill operated from May 1 to October 30, 1915, and produced 400 tons of zinc concentrate, and 200 tons of lead-iron concentrate. This represents 2.66 tons of zinc concentrate per day, and 1.33 tons of lead-iron concentrate per day.

Costs (for August, 1915).

Labor	$964.90
Power	104.85
Repairs	35.00
Fuel and extras	268.80
Total cost	1,373.55
Total cost per ton	0.915

Calculating from above concentrate assays and from the smelter contract we find (assuming average price of metals):

	Payments.
Gold	$0.95
Silver	11.00
Lead	15.50
Iron	1.85
Total	29.30
Charges	9.66
Net from sale of 1 ton of lead concentrates	19.64

	Charges.
Treatment	$4.56
Zinc	1.80
Silica	.30
Freight	2.00
Sampling	.50
Hauling to railroad	.50
Total	9.66

	Zinc payments.
Gold	$0.57
Silver	5.21
Zinc	35.90
Total	41.68
Charges	20.80
Net from sale of 1 ton of zinc concentrates	20.88

	Charges.
Treatment	$12.00
Freight	6.80
Sampling	.50
Hauling to railroad	.50
Total	20.80

Then from above:

50 tons of mill ore per day cost	$45.75
1.33 tons of Pb-Fe concentrates bring	26.18
2.66 tons of zinc concentrates bring	48.60
Profit per day	29.03

BIG FOUR EXPLORATION CO. MILL.

This mill is located at Atkinson, 7 miles below the town of Park City.

The portion of tailing deposits under lease to the Big Four Exploration Co. covers an area 3½ miles long, 500 feet wide, and averages 2 feet deep.The tonnage here available is in excess of 1,000,000 tons. Samples taken at 100-feet intervals along the flat indicate that the average metal content of this tailing is as follows: Pb, 1.8 per cent; Zn, 4.1 per cent; Ag, 3.1 ounces; Au, 30 cents; Fe, 2.5 per cent; SiO_2, 62 per cent.

The tailing is scraped up by means of 8 or 10 teams with scrapers into heaps by the side of the railroad. From these heaps the tailing is scraped over a plankway bridging the tracks and four-ton cars. A converted Cadillac touring car provided with flanged wheels acts as the motor power to a train of four dump cars of 4-ton capacity each.

The mill was operated for a few months last fall, but was closed down for the winter, during which time it has been remodeled and changed. The mill is still being altered.

FLOTATION OF ZINC SULPHIDE ORES.[a]

When the work of the Salt Lake City station began only one mill in the United States—the Butte & Superior, at Butte, Mont.—was using flotation in the recovery of zinc sulphide from slime. As the losses of zinc in slime evidently were rather large and some process that would recover this zinc was necessary, steps were taken to obtain laboratory machinery for testing the applicability of flotation for different slimes. Representatives of Minerals Separation, Ltd., the only flotation company maintaining offices in the United States at that time, did not seem to care to cooperate. The first test machines were built according to the designs printed in the only paper available, the first edition of Hoover's "Concentrating Ores by Flotation." The testing machinery described in the literature was crude and some of the first attempts at flotation at the Salt Lake City station were rather disappointing. It became necessary to visit plants where flotation was practiced before much success was obtained in the laboratory.

By that time small flotation units were being built and tested in scores of mills. In view of the drift of public interest it was decided that the most service possible could be rendered by personal visits to plants that were using or should use flotation, by distributing data, and by making suggestions. Further, much data was published in papers in technical journals. In these ways the Salt Lake City station has encouraged many people to publish their experiences. At the present time the necessity for flotation in many mills has been seen and the process installed. The only metal mining fields in which it has not been adopted to a large extent are the lead-zinc fields of Missouri, Kansas, and Oklahoma, and the Wisconsin zinc field. Here the old milling methods with jigs and tables are often in the control of men who have been slow to see the need of treating the fine slime that escapes concentration on tables. Moreover, in the mills that are under technical control in these fields the amount of slime made in any one mill under present systems of grinding is so small that there is serious doubt as to flotation plants being economical. Small flotation plants would have to be tended by the "table man" of each plant and would necessarily receive only occasional attention. Hence there is need for machines that automatically maintain froth and pulp levels and discharge tailing without attention, and also of automatic oil feeders and other accessories for plants capable of treating small tonnage—10 to 20 tons of slime a day, as determined by C. A. Wright,[b] then assistant metallurgical engineer of the Bureau of Mines.

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[a] Experimenters: G. L. Allen, F. Cameron, H. J. Morgan, C. A. Wright, and O. C. Ralston.

[b] Wright, C. A., Mining and milling of lead and zinc ores in the Missouri-Kansas-Oklahoma zinc district: Bull. 154, Bureau of Mines, 1918. 134 pp.

At present there are no machines that can practically take care of themselves for an hour at a time in a small plant with fluctuating feed. In addition there is the difficulty of training local labor to use a new process that is, to say the least, hard to understand.

In the Joplin district—Missouri-Kansas-Oklahoma field—all the ore mined on a piece of land must be treated, under the leasing system in effect, on that land, and slime from several mills can not be removed to a common point for treatment. The landowners believe that the value of a piece of ground is increased by the presence of a mill, and as the land is generally leased in 40-acre tracts, the capacity of the mills is usually restricted to 100, 200, or 400 tons a day with about 3 to 10 per cent of slime formed by crushing. A number of such mills have tried flotation but, as far as is known, few of these have yet been able to run at a profit. Although the daily loss in any one plant is not large, yet as there are over 200 mills in the Joplin district, the aggregate losses of zinc in slime is enormous. Only a change in grinding methods, the development of a small automatic flotation machine, or the changing of the present leasing system in the Joplin district will permit the general use of flotation for the slimes of the district, and the saving of the zinc now lost in the "chats." It is highly probable that the development of a small unit will be easier than a change of the leasing system.

That the flotation of the Joplin slimes is not difficult was determined by a few tests in the laboratory at the Salt Lake City station. Material containing 8.54 per cent zinc, the sample being from one of the mills treating "sheet ground" ore, could be "roughed" and cleaned in either a mechanically driven or an air machine. "Rougher" concentrates containing 35 to 49 per cent zinc could be made with most of the commercial flotation oils, such as wood creosote, coal creosote, pine oil, and pine-tar oil, pine-tar oil seeming to be the best. The recoveries were 85 to 91 per cent of the total zinc. In recleaning samples of "rougher" froth containing 39 per cent zinc, it was possible to obtain a concentrate containing 52 per cent zinc. The addition of sulphuric acid in the flotation of this ore was not necessary, but as it made the line of separation between the yellow froth and the white gangue more easily seen through the glass sides of the test machine used, amounts of about 50 pounds per ton of ore were used for that purpose. Assays of concentrates made without addition of acid were practically as good as those of concentrates made with it.

Thus there seems to be no technical difficulty in successful flotation of the zinc in the slimes of the Joplin district, although there is a serious doubt as to whether flotation can be made to pay in the present state of the art. The Bureau of Mines is not in a position to develop units of the size needed by the mills of the district, but there is evidently a good opportunity there for an inventor or a development company.

DIFFERENTIAL FLOTATION.

Differential flotation is the name given to that type of flotation in which one flotative mineral is floated in the presence of a second—the flotation of lead sulphide in the presence of zinc sulphide, for example. Some persons have called this "preferential" and others have called it "selective" flotation, but the term "preferential" has been used by the inventor of a patented process of flotation in which the ore receives a slight roast in order to deaden the surfaces of one of the flotative minerals, and the term "selective" has been applied to that type of flotation in which the oils and other flotation agents select the valuable minerals in the presence of the gangue, thus allowing the separation of a clean concentrate.

One of the most pressing problems of metallurgy at present is the treatment of complex sulphides of zinc, lead, iron, and occasionally copper. Differential flotation promises to be successful for some of these ores, and already a number of mills are using it. Two phases of the general problem are of major importance: (1) The separation of zinc and lead sulphides, and (2) the separation of zinc and iron sulphides. Possibly the latter is the more important, because the more difficult.

ZINC-LEAD SULPHIDES.

In general, mixed zinc and lead sulphides are separated by differential flotation, but usually the separation is only partial. The method in widest use is based on the fact that in a neutral or slightly alkaline solution, when an oil like pine oil is added in small enough quantity to a pulp containing both lead and zinc sulphides, galena is the only mineral that floats. Addition of more oil and acidifying the pulp causes the sphalerite to float. This method is only partly successful, and it is necessary to treat both concentrates by breaking down the flotation froth and passing the concentrates over tables in order to separate the lead in the zinc concentrate and the zinc in the lead concentrate. Even by this procedure the results are not always the most desirable, as 15 to 20 per cent of zinc is sometimes left in the lead concentrate and as much as 10 per cent lead in the zinc concentrate. Hence, differential flotation can not be regarded as a complete success. The authors know of only a few instances of better work. In fact, the authors, after running a number of tests to see if better work could be done in the laboratory, decided that there was more promise in making a mixed lead and zinc concentrate, and then separating the two metals by chemical or other means. Work on the chemical separation of such mixtures is reported in other parts of this bulletin and in another paper.[a]

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[a] Lyon, D. A., and Ralston, O. C., Innovations in the metallurgy of lead: Bull. 157, Bureau of Mines, 1918, pp. 123-169.

However, it is not impossible that better methods of differential flotation can be developed, and the Bureau of Mines intends to conduct investigations to that end.

ZINC-IRON SULPHIDE MIXTURES.

Zinc-iron sulphide mixtures offer one of the most pressing problems of differential flotation. In the past the grains of the two sulphide minerals, if too small to permit mechanical separation without fine grinding, could not be separated; nor was there any way of separating the blende and pyrite in the slime formed in the regular milling of zinc ores containing pyrite. Moreover, the two minerals can not be separated when mixed in a flotation concentration, as they have about the same specific gravity.

The only method that has proved at all successful is roasting at temperatures below 600° C., the "ignition point" of pyrite, long enough to destroy the sulphide surfaces of the pyrite particles without affecting the blende particles, followed by flotation treatment to remove the latter. The objection to this process is that in a gravity mill, where the fines are an incidental product, the fines must be dewatered, roasted, and then mixed with water for flotation.

For treating an ore that has never been milled successfully before, the installing of a process in which the ore is roasted before being mixed with water and ground for flotation is easy, for no mill product would have to be dewatered and dried before roasting, and only flotation would be used. Tests at the Salt Lake City station have shown that a simple roast to deaden the surfaces of pyrite does not suffice for coarsely ground ore, but by a longer roast the particles of pyrite are almost completely altered, much of the iron being burned to the red oxide without much oxidizing of the zinc. These tests showed, too, that sulphuric acid was almost essential in the flotation of the zinc from the roasted ore. Fine grinding after such a roast is much easier than before. This method has the advantage of a simple flow sheet for the mill. Its success, of course, is due to the fact that the roasting of zinc sulphide usually does not proceed much at a temperature below 600° C.

Addition of soluble oxidizing agents, such as potassium or other permanganates, bichromates, and chloride of lime, has been proposed by numerous inventors, but so far has been found of little value for most of the ores tested in the Salt Lake City laboratory. Usually the inventors propose to add these agents to the mill pulp in which the zinc and iron sulphides are suspended and to allow a sufficient time of contact before flotation. Whether the surfaces of one of the minerals is really oxidized seems debatable, as with mixtures of galena and blende the chromates cause the zinc mineral to float and the permanganates cause the lead mineral to float differentially. It is argued that if oxidation is all that takes place the two reagents should produce the same effect. Be that as it may, oxidizers seem to be among the most effective reagents for obtaining differential flotation.

In a number of flotation mills the zinc concentrate seems little contaminated with iron. In these mills either the pyrite has been naturally oxidized to a condition where it does not tend to float, or the slime (from a gravity mill) does not contain as much iron as the heading, because the pyrite slimes less than the galena and the blende.

In a few instances, mostly tests on a laboratory scale, copper sulphate as an addition agent improved slightly the flotation of zinc sulphide from slimes containing some iron pyrite. Copper sulphate reacts with zinc sulphide, forming a coating of copper sulphide over the zinc and this is supposed to make the zinc sulphide more easily floatable. Although copper sulphate is excellent for correcting the poor flotation of zinc sulphide, it has not proved very acceptable for differential separation in at least one mill, although laboratory tests seemed to promise improved work.

A list of the most important American and British patents covering differential flotation is as follows:

United States patents: H. A. Wentworth, No. 938732, Jan. 2, 1909; E. J. Horwood, No. 1020353, Feb. 16, 1909; A. S. Ramage, No. 949002, July 2, 1909; E. H. Nutter and Henry Lavers, No. 1067485, Sept. 1, 1911; H. H. Greenway and A. H. P. Lowry, No. 1102738, May 17, 1913; E. J. Horwood, No. 1108440, Dec. 4, 1913; Henry Lavers, No. 1142821, May 28, 1914; T. M. Owen, No. 1157176, Feb. 27, 1914; G. S. Meikle, No. 1182290, Oct. 9, 1914; R. F. Bacon, Nos. 1198589 and 1197590, June 28, Nov. 10, 1915; F. J. Lyster, Nos. 1203372, 1203373, 1203374, and 1203375, May 8, 1913.

English patents: H. L. Sulman and H. F. K. Picard, No. 8650, Apr. 9, 1910; Henry Lavers, No. 23870, Oct. 14, 1910; H. H. Greenway, No. 11471, May 16, 1913; F. J. Lyster, No. 11939, May 22, 1913; Minerals Separation (Ltd.), No. 16141, July 12, 1913; Leslie Bradford, No. 21104, Sept. 18, 1913; Amalgamated Zinc (De Bavay's) (Ltd.), Ore Treaters, No. 9049, Apr. 9, 1914; Minerals Separation and De Bavay's Processes Australia Proprietary (Ltd.), Metallurgists, Nos. 19373 and 19374, Sept. 2, 1914; Leslie Bradford, No. 21880, Nov. 2, 1914; Minerals Separation (Ltd.), Metallurgists, Nos. 5650, 8746, and 10478, Apr. 14, June 14, July 19, 1915.

At the plant of a leasing company, which is working over the tailing of the concentration mill at the Midvale (Utah) smelter of the United States Smelting Co., a mixed concentrate of zinc and lead was separated from an ore, which also contains some pyrite, by the use of 8 pounds of lime per ton of ore. The pyrite was said to be dropped from the flotation froth, and the mixed concentrate of sphalerite and galena was separated on concentrating tables. While marketable zinc concentrate can be separated and likewise marketable lead concentrate, the separation is none too clean and leaves much to be desired.

DRY CONCENTRATION PROCESSES.

There are three accepted dry methods of separating zinc minerals or zinc compounds from zinc sulphide ores, as follows: "Igneous" concentration, electrostatic separation, and magnetic separation.

IGNEOUS CONCENTRATION.

By igneous concentration is meant heating zinc ores with a blast of air to a temperature high enough to cause the zinc to volatilize and later be oxidized by the oxygen of the blast. This method is largely used in treating oxidized zinc ores and roasted sulphide ores. The usual procedure is to roast the sulphide ores to the oxide condition, mix them with fuel, such as anthracite or coke breeze, and place the mixture on a grate through which a blast of air is forced. The burning of the fuel heats the ore to a temperature at which the zinc can be reduced by the carbon or the carbonaceous gases. The zinc distills and is oxidized in the stream of air, the white zinc oxide produced, known as "American process" zinc oxide, being caught in a bag house. A discussion of igneous concentration and an account of some experiments made by the authors will be found in a subsequent section of this book.

The sulphide zinc ores now sent to igneous concentration plants are usually complex sulphides of zinc, iron, and lead which can not be separated by the standard methods of separation; hence the prices paid for these ores are usually lower than what would be paid for the several minerals if these could be mechanically separated. Many complex mixtures of zinc and iron sulphides not mechanically separable are used, the iron not interfering with the recovery of the zinc. Any lead present forms lead sulphate, which as fume is caught with the zinc oxide in the bag house, the white pigment being then known as a "leaded zinc" pigment. All of these sulphide ores are usually roasted and the sulphur dioxide gases are used for sulphuric acid manufacture. Consequently those igneous concentration plants that accept sulphide ores prefer the ores to contain some pyrite, because of its facilitating the preparation of gases suitable for making sulphuric acid. Direct "blasting" of zinc sulphide ore in making zinc oxide pigment is not in use in any part of the world, so far as the authors are informed. It is said that if sulphide ores are blasted without previous roasting, the pigment will contain too much zinc sulphate, which is undesirable. Also the zinc oxide absorbs sulphur dioxide with some avidity, which causes "livering" of the paints prepared from the product. Such material needs to be calcined at a temperature high enough to drive off the sulphur dioxide. As regards the preparation of a zinc oxide product suitable for distillation of zinc, the presence of zinc sulphate and sulphur dioxide would probably not be objectionable. Hence, as a method of concentrating the zinc in a mixed sulphide ore, igneous concentration is practicable.

At Florence, Colo., the River Smelting Co. has recently installed a blast furnace for direct treatment of mixed sulphide ores. A strong blast is used and a shallow bed; thus the furnace operates with a hot top. Moxt of the pyrite of the ore oxidizes and the iron oxide enters the slag. Some of the iron and all of the copper go into a matte, which also collects the precious metals. The lead and most of the zinc are distilled as fume and are caught in the bag houses. Some of the zinc goes into the matte and the slag, and to date the recoveries of zinc in the fume have been lower than is desirable. Also, fume deposits as crusts on the walls of the furnace, necessitating a shutdown every 10 days or two weeks for cleaning the walls.

At the Tooele smelter in Utah an attempt to smelt complex sulphides and then "blow" them in a copper converter met the same difficulty, accretions of zinc oxide forming on the walls of the converter.

ELECTROSTATIC SEPARATION.

Electrostatic separation, as previously mentioned, is one of the processes used for separating mixtures of sulphides of the same specific gravity that are not amenable to gravity concentration. Zinc ore mixtures of pyrite and sphalerite in coarse crystals are often treated by this process. Such ore needs only drying before treatment on the electrostatic separator. The principle of the electrostatic separators need not be explained here, as there is ample literature on the subject. From 1905 to date there have usually been several papers published every year on this process in such publications as Mineral Industry, Engineering and Mining Journal, Metallurgical and Chemical Engineering, and the Transactions of the American Electrochemical Society. The nonconductive mineral particles pass through the separator inert, but the particles of the conductive minerals acquire an electrical charge and are repelled from one electrode. Sphalerite is a poor conductor or a nonconductor, and pyrite is a good conductor. Galena is likewise a good conductor and accompanies the pyrite. Fine dust can not be treated on such a machine, its use being limited to the treatment of "sands."

As ores vary widely in their electrical properties, the only way of finding whether a given ore of zinc sulphide and other sulphides can be separated by this method is to test the ore in a commercial machine. With some ores electrostatic separation produces concentrates of commercial grade without further treatment, but as a rule none of the electrostatic processes will make a clean separation of the average complex sulphide ore found in the intermountain States. The fact that pyritic ore does not have to be roasted before electrostatic separation is important, as the sulphur can be saved, if desired. If the magnetic separation process were used, some of the sulphur in the ore would be burned off and escape in the smelter gases, forming a nuisance. Another advantage of the electrostatic method is that mixtures of galena, pyrite, chalcopyrite, and sphalerite can be separated into a product containing the first three minerals mentioned and a second product containing the sphalerite. These two products can both be marketed to good advantage. The magnetic separation process, however, makes a product containing the iron and the copper and another product containing the lead and the zinc; and this latter mixture can not be marketed to as good advantage as if the lead and zinc minerals were separated.

MAGNETIC ROASTING AND SEPARATION.

The method of separating zinc and iron sulphides by the use of a magnetic separator is an old one. There are many ores, in districts like the Wisconsin zinc district, which are mixtures of the two sulphides that can be easily separated by this method, and in these places the practice has become standard. In the intermountain States, however, there are many sulphide ores that do not respond to this treatment. Determining the character of such ores by their appearance is difficult, and the best way of finding whether a given sample of ore is amenable to magnetic separation is to test it.

EXPERIMENTS AT SALT LAKE CITY STATION.

The Zinc Concentrating Co., in cooperation with the University of Utah and the Bureau of Mines, set up one of its Campbell magnetic separators and a small Etherington-Singer roaster at the Salt Lake City station in order to test mixed sulphide ores from various western districts.

DESCRIPTION OF ROASTER AND MAGNETIC SEPARATOR.

The Etherington-Singer roaster is a revolving cylindrical kiln, tapering slightly toward the flue end and being fired with oil or gas. The small laboratory roaster used for the tests was an iron cylinder 7 feet long and about 1 foot in diameter. Heating was by means of two large oil blow-torch flames, the roaster generally requiring about one hour's heating to attain the temperature necessary for a quick roast under these conditions. The roaster was so inclined that when it was making four revolutions per minute the ore passed through it in 7 minutes. The ore, just before discharging, showered through the flame from the torches. It was found that under these conditions enough of the sulphur in the pyrite was quickly eliminated and the product, because of the nonoxidizing character of the flame, had not been oxidized too much. Of course, the object in magnetic roasting is to raise the ore to such a temperature that the pyrite breaks up into FeS or $Fe_x S_{x+1}$, and S. Pyrite is not magnetic, but the lower sulphides of iron are ferromagnetic, and are capable of being lifted by a magnet of low intensity. Sphalerite is nonmagnetic; hence the magnetic separator removes the iron and leaves the sphalerite, with any other nonmagnetic minerals present.

The Campbell magnetic separator comprises a shaking plate so inclined that the ore travels down it in a thin sheet; above this plate are suspended three electromagnets. Beneath the face of each magnet passes a canvas belt, so that any particle attracted to the magnet is arrested by the belt, which carries the particle over the edge of the shaking table and beyond the field of the magnet, when the particle drops from the belt. The three magnets used are of increasing intensity, so that the separator makes three products which are decreasingly magnetic, and a nonmagnetic product. The distance of the faces of the magnet poles from the table is adjustable, and rheostats control the amperage flowing through each magnet, so that a stronger or weaker magnetic field is obtainable.

TESTS OF ORES.

Very few of the western ores not now being worked that were tested had sufficiently coarse crystallization to permit magnetic separation of the iron sulphide from the zinc sulphide. Tests were made of ore from the Utah-Apex mine, Bingham, Utah; zinc middling from the Broadwater mill at Park City, Utah; zinc middling from the Consolidated Nevada-Utah mill at Pioche, Nev.; ore from the Greenwood lease, Pioche, Nev.; ore from the Carlyle mine, Duncan, Ariz.; ore from the Middle Golconda mine, and ore from the C. O. D. mine, near Chloride, Ariz., and others.

As stated above, only a few of the ores responded satisfactorily in the tests. The essential data on the more important tests follows.

ZINC-IRON MIDDLING FROM UTAH-APEX MINE, BINGHAM, UTAH.

The Utah-Apex mine at Bingham, like most of the other Bingham lead mines, has not saved the zinc in its ore, much of the ore being shipped crude to the lead smelters. This ore has the following analysis:

Composition of ore from Utah-Apex mine, Bingham, Utah.

Lead (Pb)	per cent..	16.65
Zinc (Zn)	do....	14.45
Iron (Fe)	do....	17.80
Insoluble	do....	16.50
Silver (Ag)	ounces per ton..	4.34
Gold (Au)	do....	.04

The lead mineral in this ore can be separated by mechanical concentration, but the zinc and iron minerals, having the same specific gravity, are obtained as a zinc-iron middling and the high iron makes this middling unsalable as a zinc product. As the ratio of the iron content to the silica content is such that the ore is self-fluxing, it is sought by the smelters and often contracts can be obtained which ignore the high zinc content. Hence very little concentration of this grade of ore has been practiced. By making a middling on Wilfley tables, a product containing zinc sulphide, iron sulphide, and quartz was obtained, and on tabling the gangue a product of the following analysis was obtained:

Results of analysis of zinc-iron middling.

Lead (Pb)	per cent..	1.31
Zinc (Zn)	do....	34.80
Iron (Fe)	do....	12.70
Insoluble	do....	13.20
Silver (Ag)	ounces per ton..	1.61
Gold(Au)	do....	.01

Such a product could not be marketed to advantage as zinc concentrate; so this middling was subjected to magnetic roasting and separation. In the treatment on the magnetic separator, the intent was to remove most of the iron, the magnets being set for a low intensity, and then re-treat the zinc concentrates with a more intense field. The results of these tests are given in Table 7 following:

TABLE 7.—*Results of magnetic separation of Utah-Apex zinc-iron middling.*

Sample.	Weight, pounds.	Pb.		Fe.		Zn.		Insoluble, per cent.	Ag.		Au.	
		Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.		Ounce.	Per cent of total.	Ounce.	Per cent of total.
Raw heading	74.5	1.31	100.0	12.7	100.0	34.8	100.0	13.2	1.61	100.0	0.01	100.0
Roasted heading	66.0	1.43	96.8	12.9	90.2	38.1	97.2	13.8	1.63	90.2	.01	88.6
Nos. 1 and 2 "iron" middlings	5.75	.95	5.8	49.6	33.5	6.0	1.4	8.6	2.19	11.6	.01	8.7
No. 3 "iron" middling	5.75	1.55	9.4	40.1	27.0	12.1	2.8	9.4	3.11	16.6	.01	8.7
Zinc concentrate	53.0	1.55	86.8	6.1	37.8	45.0	94.9	14.6	1.51	74.0	.01	80.0
All "iron" middlings.	3.75	2.98	8.7	25.2	8.0	25.2	2.7	9.4	4.12	10.4	.02	4.2
Zinc concentrate	48.5	1.43	72.5	4.94	28.2	46.3	89.5	14.8	....	......	....	......

ORES FROM THE CONSOLIDATED NEVADA-UTAH AND THE GREENWOOD MINES, PIOCHE, NEV.

The ore from the Consolidated Nevada-Utah mine and that from the Greenwood lease are representative of two types of ore found in the Pioche, Nev., district. The former represents the deeper lying ore in the vein deposits of that district, and the latter the complex, partly oxidized sulphides nearer the surface. This latter type of ore is much more complex and difficult to separate than the first. It can be seen from Tables 8 and 9 that marketable zinc concentrate could be prepared from the Consolidated ore, whereas a good separation of the zinc and lead minerals from the iron sulphide in the Greenwood ore is practically impossible. High-iron zinc ore or zinc-lead ore is sold for the manufacture of zinc oxide or leaded zinc oxide, although the price paid for the ore is not equivalent to that paid for the separate lead and zinc concentrates that might be made. The companies buying such ore usually specify 35 per cent combined zinc and lead content as the lower limit of the material that they will accept for making zinc oxide. The Greenwood ore can be prepared for such a market by double treatment with magnetic separators, as can be seen from Table 9, but the loss of zinc in the iron concentrate is too great.

TABLE 8.—*Results of magnetic separation of ore from Consolidated Nevada-Utah mine.*

Sample.	Weight, pounds.	Pb.		Fe.		Zn.		Insoluble, per cent.	Ag.		Au.	
		Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.		Ounces.	Per cent of total.	Ounces.	Per cent of total.
Raw heading	100.0	2.92	100.0	28.4	100.0	17.9	100.0	7.8	11.46	100.0	0.22	100.0
Roasted heading	84.0	3.27	94.3	30.3	91.2	19.0	89.2	9.2	12.52	92.0	.24	91.6
No. 1 "iron" middling	5.0	1.72	3.4	53.0	11.3	5.6	1.9	5.0	12.32	6.4	.28	7.6
No. 2 "iron" middling	7.75	1.43	4.0	50.5	15.1	6.2	3.0	4.0	14.38	10.6	.22	8.4
No. 3 "iron" middling	31.5	2.47	28.3	43.5	53.0	9.5	18.7	6.6	15.82	47.4	.38	59.7
Zinc concentrate	37.5	5.05	68.8	13.6	19.7	35.5	83.3	12.6	11.14	39.7	.24	44.7
No. 3 "iron" reseparated, making—												
Nos. 1 and 2 "iron" middlings	7.5	1.37	3.7	51.0	14.8	5.4	2.5	4.0	11.58	8.2	.36	13.4
No. 3 "iron" middling	20.5	2.32	17.3	45.3	35.9	7.9	10.1	6.4	16.06	31.3	.32	32.6
Zinc concentrate	3.5	4.94	6.3	21.6	2.9	26.1	5.7	13.4	16.86	5.6	.36	6.3
No. 1 zinc concentrate reseparated, making—												
All "irons"	4.5	3.57	5.8	28.0	4.9	17.0	4.8	13.8	13.84	5.9	.56	12.5
Zinc concentrate	31.5	4.28	49.2	11.1	13.5	38.1	75.3	12.0	8.70	26.1	.46	71.9
Last zinc concentrate reroasted and reseparated, making—												
No. 1 "iron"	3.0	2.14	2.3	45.9	5.3	7.8	1.5	5.8	4.26	1.2	.16	2.4
No. 2 "iron"	1.5	4.88	2.7	23.0	1.3	21.8	2.0	14.4	13.72	2.0	.96	7.1
Zinc concentrate	24.0	5.23	45.8	6.5	6.0	44.6	67.1	12.2	8.72	19.8	.38	45.3

TABLE 9.—*Results of magnetic separation of ore from Greenwood lease.*

Sample.	Weight, pounds.	Pb.		Fe.		Zn.		Insoluble, per cent.	Ag.		Au.	
		Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.		Ounces.	Per cent of total.	Ounces.	Per cent of total.
Raw heading	44.0	8.21	100.0	15.1	100.0	20.0	100.0	25.2	8.11	100.0	0.04	100.0
Roasted heading	35.5	9.05	88.9	15.7	84.2	20.0	80.8	27.4	9.06	87.7	.04	80.7
Nos. 1 and 2 "iron" middlings	1.0	5.77	1.8	44.9	8.0	5.2	.7	12.6	8.28	2.6	.06	4.2
No. 3 "iron" middling	2.75	8.56	7.4	35.8	17.6	7.1	2.7	17.0	10.04	8.6	.06	11.6
Zinc concentrate	29.5	9.51	87.6	13.6	72.0	21.7	90.2	28.2	8.98	82.3	.04	83.1
"Iron" middling from zinc concentrate	17.0	9.28	49.3	15.7	47.8	18.8	45.0	29.8	8.90	46.9	.04	47.9
Re-treated zinc concentrates	9.0	9.75	27.3	8.7	13.6	27.4	34.8	26.6	8.20	22.9	.04	25.4
No. 2 "iron" middlings from No. 1 "iron"	8.5	8.45	22.4	18.9	29.9	15.0	18.0	28.8	8.88	19.7	.04	23.9
Zinc concentrate from "iron" middlings	5.0	8.50	13.2	8.66	7.8	25.5	18.0	29.8	7.48	11.6	.04	14.1

ZINC-IRON MIDDLING FROM BROADWATER MILL, NEAR PARK CITY, UTAH.

Near Park City, Utah, a large accumulation of old mill tailing is being treated by a number of mills, and large amounts of zinc-iron middling are produced. The question has been raised as to what should be done with it. The sample from the Broadwater mill represents a product such as could have been made during the milling of countless tons of the Park City ores in the past, and could be prepared from the ores now being mined. A marketable zinc concentrate from such zinc-iron middling is possible, as shown by the tests reported in Table 10 following:

TABLE 10.—*Results of magnetic concentration of zinc middling from the Broadwater mill.*

Sample.	Weight, pounds.	Pb.		Fe.		Zn.		Insoluble, per cent.	Ag.		Au.	
		Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.		Ounces.	Per cent of total.	Ounces.	Per cent of total.
Raw heading	112.0	6.36	100.0	27.4	100.0	18.1	100.0	3.2	12.74	100.0	0.02	100.0
Roasted heading	98.0	......	......	......	......	......	......	......	......	......	......	......
Nos. 1 and 2 "iron" middlings	19.0	4.23	11.8	53.4	33.1	3.3	3.1	1.8	14.97	19.9	.03	25.5
No. 3 "iron" middling	28.5	3.93	15.7	44.7	41.5	5.0	7.1	1.8	16.14	32.2	.02	25.5
Zinc concentrate	50.5	8.95	63.5	11.6	19.1	36.9	92.8	4.8	12.82	45.8	.02	45.2
Zinc concentrate reseparated making—												
All "iron" middlings	9.0	6.01	7.6	37.8	11.1	8.0	3.6	2.4	18.06	11.8	.02	8.0
Zinc concentrate	40.5	9.65	55.0	6.8	9.0	41.7	83.7	4.8	11.36	32.2	.02	36.2

ORE FROM CARLYLE MINE, STEEPLEROCK, N. MEX.

The ore from the Carlyle mine, near Steeplerock, N. Mex., was tested without removal of the gangue. This was a complex sulphide ore carrying zinc, lead, copper, and iron. The deposit at this mine represents a fairly common ore type in this district, the sulphide ore coming from beneath an old bonanza oxidized gold-ore body. There are many similar mines in which oxidized ores of silver, gold, or copper changes to complex-sulphide ores below the water level. In this sample the sulphides were more or less separable from each other but were intimately combined with the gangue, so that the preparation of products not contaminated with too much gangue was difficult. This was especially true of the zinc concentrate. The only encouraging feature of the test was that the iron products from the magnetic separation carried a good proportion of the copper of the original ore in a concentrate that was marketable. (See Tables 11 and 12.)

The raw ore was ground to pass 20-mesh and then was given a heavy roast in the laboratory roaster. The calcines were then put through the magnetic separator with the magnets set fairly close. The "irons" from the above test were reground to pass 40-mesh and then put over the magnetic separator again, making a magnetic and a nonmagnetic product. The zinc concentrate obtained from the roasting and separation of the Carlyle zinc middlings from a table test was treated on the small test table to see what kind of a separation could be obtained after roasting. The results were as follows:

TABLE 11.—*Results of magnetic separation of ore from Carlyle mine, roasting without tabling.*

Sample.	Weight, pounds.	Pb.		Zn.		Fe.		Cu.		Insoluble, per cent.
		Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.	
First separation:										
Roasted heading	19.0	6.43	100.0	7.6	100.0	3.8	100.0	1.21	100.0	69.6
All "iron" middlings	2.25	13.32	24.6	6.8	10.6	10.0	31.2	6.74	66.1	47.8
Zinc concentrate	16.5	4.76	64.4	7.8	89.4	2.6	59.5	.50	35.6	73.8
Separation of iron concentrate:										
Heading	845.0	11.32	100.0	6.8	100.0	10.0	100.0	6.74	100.0	47.8
Nonmagnetic	390.0	11.2	38.9	8.3	55.3	3.6	16.6	.88	6.0	61.6
Magnetic	395.0	12.97	45.7	6.0	41.2	16.7	78.2	11.92	82.8	32.6
Tabling zinc concentrate from zinc middling:										
Heading	1,000.0	9.2	100.0	17.9	100.0	3.3	100.0	.38	100.0	39.2
Lead concentrate	245.0	13.7	36.5	25.7	35.2	3.0	22.3	.50	22.1	15.0
Zinc concentrate	350.0	9.64	36.6	21.5	42.1	3.2	33.9	.44	40.5	38.0
Tailing	350.0	4.75	18.1	10.2	19.9	3.5	37.1	.44	40.5	67.4

In another test the ore was first tabled after being ground to pass a 20-mesh screen. Three products were obtained—a lead concentrate, a zinc middling, and a tailing. The slime was not recovered; this accounts for the low recovery of copper, as the chalcopyrite slimes badly and was lost. The zinc middling obtained was put through the laboratory roaster and then through the magnetic separator. The concentrate obtained was again sent through the magnetic separator, with the magnets set closer together, to see if any more iron could be removed. The results of the tests are as follows:

TABLE 12.—*Results of magnetic separation of ore from Carlyle mine, with tabling.*

Sample.	Weight, pounds.	Pb.		Zn.		Fe.		Cu.		Insoluble, per cent.
		Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.	Per cent.	Per cent of total.	
Table test:										
Heading	40	14.2	100.0	9.4	100.0	5.0	100.0	1.16	100.0	37.6
Lead concentrate	5	62.0	54.8	12.0	17.3	3.3	8.3	1.38	14.9	3.2
Zinc middling	9.75	12.5	21.5	18.4	47.6	5.4	26.3	2.21	46.3	44.4
Tails	19.4	2.02	6.9	4.1	21.1	3.6	35.0	.55	23.1	32.6
Zinc middling roasted:										
Raw heading	3,626	12.5	100.0	18.4	100.0	5.4	100.0	2.21	100.0	44.4
Roasted heading	3,977	10.1	89.5	18.0	95.2	5.6	100.0	1.93	95.4	38.2
Through separator	3,862	10.1	100.0	16.0	100.0	5.6	100.0	1.93	100.0	38.2
No. 1 and No. 2 "iron" middlings.	255	11.7	7.6	4.3	1.8	25.1	29.6	12.21	41.8	18.0
No. 3 "iron" middling	285	12.3	9.0	9.5	4.4	12.5	16.5	7.18	27.5	33.2
All "iron" middlings	195	19.3	9.6	14.9	4.7	6.5	5.9	2.21	5.8	35.2
Zinc concentrate	3,070	9.2	72.5	17.9	89.0	3.3	47.0	.38	15.7	30.2

ORES FROM MINES NEAR CHLORIDE, ARIZ.

The ore from the C. O. D. mine, near Chloride, Ariz., is from a deposit somewhat similar to that in the Carlyle mine. However, this sample yielded commercial grades of concentrate, both of lead and of zinc, as shown in Table 13.

TABLE 13.—*Results of magnetic separation of ore from C. O. D. mine, near Chloride, Ariz.*

Sample.	Weight, pounds.	Pb.		Zn.		Fe.		Insoluble, per cent.	Ag.		Au.	
		Per cent.	Per cent recovered.	Per cent.	Per cent recovered.	Per cent.	Per cent recovered.		Ounces.	Per cent recovered.	Ounces.	Per cent recovered.
Tabling test:												
Ore	....	3.69	......	7.0	......	7.1	......	61.6	18.5	......	0.30	......
Table feed	47.5	3.57	100.0	7.2	100.0	6.9	100.0	62.6	16.9	100.0	.30	100.0
Lead concentrate	2.5	39.70	58.6	7.3	5.6	17.2	18.1	2.4	64.5	20.2	1.40	24.5
Zinc concentrate	13.37	2.50	19.5	20.0	81.7	13.7	53.8	20.8	27.4	45.6	.45	42.3
Tailing	27.50	.24	3.9	1.5	12.7	2.4	20.1	89.2	5.5	18.8	.05	9.8
Slime	4.20	......	......	......	......	......	......	......	20.3	10.6	.30	8.4
Roasting and separating first zinc concentrate:												
Raw heading	200	2.5	100.0	20.0	100.0	13.7	100.0	20.8	27.4	100.0	.45	100.0
Roasted heading	176	2.8	98.8	24.8	100.0	15.0	96.5	21.6	29.5	95.1	.45	88.2
Nos. 1 and 2 "iron" middlings	34	1.13	7.9	3.0	2.3	40.7	52.2	10.8	51.9	35.2	.90	40.0
No. 3 "iron" middling	24	2.26	10.9	7.9	4.4	29.4	26.7	18.6	32.2	15.4	.50	15.7
Zinc concentrate	110	3.	74.2	38.5	97.1	3.8	18.9	26.0	20.8	45.6	.25	35.9
Last zinc concentrate tabled:												
Feed	105	3.33	100.0	38.5	100.0	3.8	100.0	26.0	....	......	....	......
Lead concentrate	4	20.0	22.8	37.7	3.7	3.5	3.5	7.3	....	......	....	......
Zinc concentrate	65	2.86	58.2	44.2	71.0	3.5	57.2	19.1	....	......	....	......
Zinc middling	34	1.55	15.1	27.0	22.7	4.6	39.1	41.7	....	......	....	......

The Middle Golconda mine, near the Chloride, Arizona, district has zinc-iron sulphide ore carrying small amounts of silver, lead, and copper. On account of the small lead content it was difficult to get a clean galena concentrate free from zinc on the concentrating tables. The zinc-iron middling could be separated easily into a zinc concentrate of commercial grade and an iron concentrate, containing some lead and silver, which could either be combined with the lead concentrate produced on the tables, or discarded, according to the distance from a lead smelter. The flotation of the slime of this ore produced a mixed lead-zinc sulphide concentrate that could be marketed, although not to as good advantage as separate lead and zinc concentrates. The results are shown in Table 14.

TABLE 14.—*Results of concentration tests of ore from Middle Golconda mine.*

	Weight.	Pb.		Zn.		Fe.		Cu.		Insoluble, per cent.	Ag.		Au.	
		Per cent.	Per cent recovered.	Per cent.	Per cent recovered.	Per cent.	Per cent recovered.	Per cent.	Per cent recovered.		Ounces.	Per cent recovered.	Ounces.	Per cent recovered.
Table test of 20 to 40 mesh ore:	*Ounces.*													
Ore	....	0.80	....	14.0	....	9.7	....	....	....	48.2	3.47	....	0.04	....
Heading	336	.83	100.0	12.4	100.0	9.2	100.0	0.11	100.0	53.0	1.96	100.0	.02	100.0
Lead concentrate	4	34.20	49.4	23.1	2.2	10.5	1.4	.36	3.8	3.4	17.06	10.3	.28	16.7
Zinc concentrate	68	2.19	53.4	35.3	57.6	11.2	24.6	.33	60.5	15.0	....	....	....	....
Zinc middling	88	.18	5.4	12.5	26.4	9.8	28.0	.17	41.0	49.6	....	....	....	....
Tail	161	.12	6.8	2.6	10.0	8.1	42.7	.08	3.5	70.6	.89	21.7	.01	23.8
Table test of 40 to 80 mesh ore:														
Heading	160	1.48	100.0	16.2	100.0	9.3	100.0	.14	100.0	46.2	2.46	100.0	.02	100.0
Lead concentrate	4	31.20	52.8	24.5	3.8	12.1	3.2	.14	2.7	1.6	....	....	....	....
Zinc concentrate	37	3.84	59.0	41.5	59.2	10.8	26.8	.33	54.5	6.0	....	....	....	....
Zinc middling	21	.41	3.6	23.0	18.7	10.4	14.6	.08	7.6	33.0	....	....	....	....
Tailing	77	.03	1.0	2.2	6.5	7.8	40.2	Tr.	....	70.4	.67	13.1	.01	24.0
Roasting and separating zinc concentrate from above:	*Grams.*													
Raw heading	1,870	2.19	100.0	41.5	100.0	11.0	100.0	.28	100.0	6.2	....	....	....	....
Roasted heading	1,740	2.01	85.7	42.0	94.1	11.0	92.8	.39	100.0	7.6	....	....	....	....
Dust	120	5.33	10.3	16.5	2.7	7.4	4.6	.99	17.5	36.6	....	....	....	....
First and second iron middlings	114	1.48	4.8	6.5	1.0	30.7	18.3	1.41	23.7	18.6	7.86	....	.06	....
Third iron middling	72	2.96	6.1	17.5	1.7	26.7	10.0	3.70	31.8	13.6	17.28	....	.22	....
First zinc concentrate	....	1.95	....	47.2	....	8.2	....	.11	....	6.6	....	....	....	....
Second and third iron middlings	50	3.61	8.3	25.7	2.8	24.8	10.3	.44	5.1	10.6	13.42	....	.14	....
Second zinc concentrate	1,415	1.42	58.6	49.5	96.0	7.7	57.0	.17	35.4	6.0	....	....	....	....
Flotation of slime:														
Slime heading	500	1.72	100.0	14.1	100.0	8.8	100.0	.05	100.0	43.0	4.98	100.0	.06	100.0
Concentrate	182	4.62	98.8	29.6	77.2	10.9	45.4	.11	80.0	17.6	....	....	....	....
Tailing	302	.05	1.7	6.5	27.8	8.8	60.0	.05	60.0	60.2	1.23	14.9	.01	10.0
Concentrate	74	11.30	97.3	33.5	35.2	9.3	15.6	.61	32.0	15.4	....	....	....	....
Tailing	422	.12	5.9	11.0	65.9	.7	92.7	.05	84.0	46.0	2.04	34.6	.02	26.7

SUMMARY OF THE MAGNETIC-SEPARATION TESTS.

A number of the less complex mixed-sulphide ores from mines in the western States have been tested for magnetic separation of the iron sulphide with varying degrees of success. Some of them give commercial grades of concentrate.

ROASTING ZINC SULPHIDE ORES.[a]

CHLORIDIZING ROASTING.

For some years previous to the inception of the work at the Salt Lake City station, various chloridizing processes had been popular in Utah. The large supplies of salt available in this State made chloridizing roasting and brine leaching a logical development in the metallurgical treatment of low-grade and complex ores. The plants of the Mines Operating Co. at Park City and of the Knight-Christensen Metallurgical Co. at Silver City were designed to remove copper, gold, and silver from the low-grade siliceous ores found in many of the western mining districts. Recovery of the lead, in addition to these three metals, had not received much attention, although some lead had been recovered in these plants. Therefore, the Salt Lake City station made a study of the action of the lead and zinc minerals during chloridizing-blast-roasting and leaching, as practiced in the above mills, with the object of determining whether the processes could be extended or modified in order to recover all of the lead and the zinc.

The results of the work on lead have been published by the Bureau of Mines in Bulletin 157.[b] Seemingly, it should be easily possible to modify the processes used in order to make proper recovery of the lead.

The work with the zinc minerals was unsuccessful as only part of the zinc could be chloridized. The question of chloridizing oxidized zinc ores is discussed in another part of this bulletin (p. 120). As regards the zinc-sulphide ores, satisfactory roasting and chloridizing in either the Holt-Dern shaft roaster or the Knight-Christensen downdraft roaster seemed impossible. With an ore containing 5 per cent of zinc as the sulphide, in a quartz gangue, it was found that under the conditions prevailing in the blast roasters the zinc sulphide did not have time to roast, either in the presence or absence of salt. None to 12 per cent of the zinc would volatilize, probably as chlorides, and 5 to 35 per cent of the zinc in the calcine was soluble. Leaching with acidified brine would usually extract not more than half of the zinc in the calcine, including the water-soluble or chloridized zinc.

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[a] Experimenters: W. G. Woolf, C. L. Larson, M. J. Udy, Walter Neal, R. W. Johnson, and J. F. Cullen.

[b] Lyon, D. A., and Ralston, O. C., Innovations in the metallurgy of lead: Bull. 157, Bureau of Mines, 1918 176, pp.

To describe in detail the various tests made on this material would serve no useful purpose. The following brief summary will suffice:

The ore was roasted after being ground to many different sizes, varying from 8-mesh to 100-mesh material. The quantity of fuel used varied from 1 to 5 per cent, and the amount of salt from 3 to 10 per cent of the weight of the ore. The blast pressure was varied in order that temperatures of 600° to 900° C. could be obtained. The roasting period was varied from 15 minutes to 3 hours. In no test did it seem possible to roast more than 50 per cent of the zinc to a form soluble in acid brine. The chemical and physical properties of zinc sulphide are such that it usually roasts very slowly, and the brief time that the mineral is heated in an oxidizing atmosphere in the blast roasters used does permit complete roasting. With the shallow bed on the down-draft roaster, which gives a 15-minute roast, usually not more than 20 per cent of the zinc is oxidized or chloridized, and in many of the tests the total amount of zinc removed during such treatment was only about 5 per cent.

OXIDE ROASTING.

By oxide roasting is meant the roasting of zinc-sulphide ores to as low a percentage of sulphur as possible. As zinc sulphide is one of the most refractory of the major-metal sulphides to roast, advancement in the art of roasting these ores is far from satisfactory. However, no doubt considerable improvements will be made in mechanical contrivances for obtaining continuous roasting of such ores. The chief objection to existing types of furnaces is that when the roast is carried to the point of satisfactory elimination of sulphur the costs of building and operation are very high. For zinc-retort smelting it is important that no sulphur be left in the calcine. The statement has often been made that every pound of sulphur left in the calcine, restrains 2 pounds of zinc from distilling in the retorts.

For leaching the zinc calcine with sulphuric acid, complete desulphurization of the ore is not required but complete oxidation of the sulphides of zinc is necessary. In other words, the amount of zinc converted to sulphate of zinc is immaterial, as long as all of the sulphide of zinc is roasted. Usually the lower the temperature of roasting the greater is the percentage of zinc sulphate formed, but also the greater is the liability of leaving some unroasted zinc sulphide unless the time of roasting is extended over a long period. The higher the temperature of roasting the higher is the percentage of oxide of zinc formed, because zinc sulphate decomposes thermally at temperatures above 680° C. The desirability of making zinc sulphate will depend on the acid requirements of the ore during leaching. An electrolytic zinc process returns the used acid for leaching and only enough new acid need be added to bring the solution up to the desired strength and make up for mechanical losses.

SULPHATE ROASTING.

The principles of sulphate roasting of zinc sulphide in the ordinary reverberatory type of roaster are fairly well known, but the sulphating efficiency of blast roasters has not been generally tested. As chloridizing roasting had failed in the tests just described, sulphating roasting did not seem to be a very promising field for experiment. However, a few tests were made in the Holt-Dern laboratory roaster, running slowly and with a low blast. The low blast permitted a low temperature and a long roasting period, conditions which are necessary when attempting to form zinc sulphate. The quantity of water-soluble zinc after such a roast was nearly always small and rarely exceeded 10 per cent of the total zinc content. Hence the blast roaster is not adapted to such work.

In a series of oxidizing roasts of zinc-sulphide ores, made in a small gas-fired reverberatory roaster, many of the roasts were leached with water. The conditions of formation of zinc sulphate in a muffle are discussed in connection with that series of tests.

At Great Falls, Mont., the Anaconda-Wedge roaster is giving a 22-hour roast to complex sulphide ore, the temperature being about 600° C., and the ore being rabbled four times a minute. About 30 to 50 per cent of the zinc is sulphated. Marmatite ores give better results in sulphate roasting than "free" ores, possibly because of the catalytic effects of iron oxide in direct contact with the zinc oxide. The low temperature used is to prevent the breaking up of zinc sulphate and also the formation of ferrites of zinc.

An extensive series of roasting tests of various complex-sulphide ores containing zinc was made at varying temperatures and for different lengths of time to determine the best conditions for roasting such ores. At first a gas muffle furnace was used, the ore being placed on the bottom of the muffle. A platinum and platinum-rhodium thermocouple protected by alundum tubing was used to measure the temperatures in the furnace. It was soon found that roasting in such a furnace was of no value, as the ore next the bottom of the muffle became much hotter than that farther from the bottom. Hence the use of roasting dishes became necessary, and a gas reverberatory roasting furnace 2 feet by 3 feet in size, was used, to parallel more nearly the conditions in a commercial roaster.

The reason for studying roasts of this kind was that considerable trouble was then being experienced in the experimental electrolytic zinc plants at Anaconda, Mont.; Trail, B. C.; Murray, Utah; Welland, Ontario, and other places, in getting a calcine that was completely roasted. It seemed to be the common experience that in such calcines 60 to 70 per cent of the zinc would be acid-soluble;

the remainder was suspected to be largely sulphide of zinc, although ferrite of zinc was also known to be present. Chemical analyses showed sulphur left in the calcine. The question for determination was whether it would be possible, even by roasting in shallow dishes in a furnace of controlled temperature and with considerable rabbling, to calcine these ores in a short length of time, as can usually be done with copper ores in furnaces of this type. For instance, the Wedge zinc roaster can usually give the ore not more than 8 hours' roast, whereas the old Hegeler furnaces have been used for as much as a 60-hour roast. The more recent Anaconda-Wedge furnace gives a 20 to 22 hour roast.

A further point of interest was to determine the relative rates of roasting of lead and zinc sulphides in a mixed ore, as lead leaching was being tested at the same time in this laboratory. By treating the calcine with hot water the zinc sulphate formed could be leached and determined. In a similar way the total oxidized zinc could be determined by treatment with a solution of acetic acid (sulphuric acid tends to dissolve zinc sulphide slowly). The solution of acetic acid also leaches much of the oxidized lead. A solution of ammonium acetate dissolves lead sulphate, as does a saturated solution of sodium chloride. Finally, for practical purposes, a sulphuric acid leach to recover the total soluble (oxidized) zinc, followed by a brine leach to get the oxidized lead, was made.

ROASTING OF A LOW-IRON ORE FROM DALY-JUDGE MINE.

DESCRIPTION OF ORE AND OF METHOD USED.

The first ore to be roasted, known as sample 1, was from the Daly-Judge mine, of Park City, Utah. This sample represents a great deal of the Park City ore; it contained 16.88 per cent Zn, 14.87 per cent Pb, 2.25 per cent Fe, 10 per cent CaO, 30.8 per cent insoluble, and 12.7 per cent S. The sample was crushed to pass a Tyler standard 10-mesh screen and then gave the following screen analysis:

Results of screen analysis of Daly-Judge No. 1 ore crushed to pass a 10-mesh screen.

	Per cent.
On 20-mesh	21.8
On 35-mesh	19.8
On 65-mesh	15.0
On 100-mesh	8.8
On 150-mesh	5.4
On 200-mesh	6.2
Through 200 mesh	22.0

The curves in figure 2 show the percentages of the zinc and of the lead that were oxidized during the roasting of portions of this sample for different lengths of time at different temperatures. The roasts were made in duplicate and the average of the two results taken. Oxidized zinc and lead were determined as previously described. The portions roasted weighed 300 grams each and were placed on clay roasting dishes 6 inches in diameter. At the end of each hour they were withdrawn, stirred, sampled, and returned to the furnace. The depth of the bed of ore being roasted was about 1 inch.

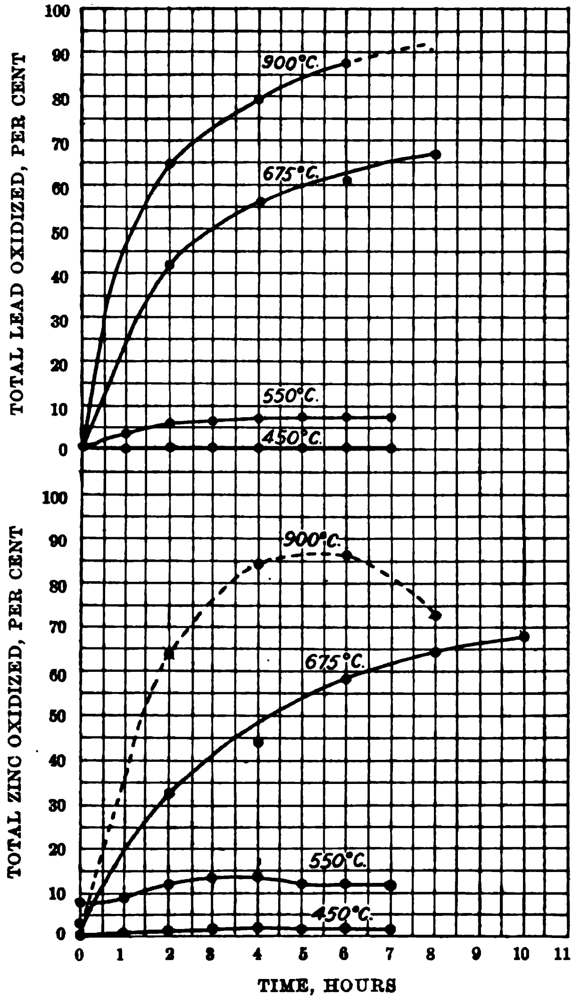

FIGURE 2.—Curves showing effect of temperature on speed of roasting, Daly-Judge No. 1 sample, 10-mesh size.

DISCUSSION OF RESULTS OF TEST.

The results show that 450° C. is too low a temperature for roasting either the sphalerite or the galena. At 550° C. little roasting takes place; at 675° C. roasting proceeds much more rapidly so that as much as 70 per cent of the zinc can be roasted in 10 hours, the greatest length of time that material can be held in the usual Wedge or McDougall type of roaster. Probably, more continuous rabbling would permit faster roasting at this temperature. The curve for the recovery of zinc at 900° C. is dotted to distinguish it from the curves for the lower temperatures; for this curve a 10 per cent solution of sulphuric acid was used to determine the oxidized zinc in place of a 40 per cent solution of acetic acid. The sulphuric acid solution seems to dissolve some zinc sulphide as well as the oxidized zinc.

Perhaps the most striking feature brought out in figure 2 is that the speed of roasting both sphalerite and galena depends largely on the temperature; also, these two minerals seem to roast at about the same rate of speed. The drop toward the end of the curve for zinc recovery at 900° C. is probably due to the formation of zinc ferrites or similar compounds, which are known to form at such a temperature. In fact, in roasting an ore containing much iron, past work has shown repeatedly that 750° C. is about the highest temperature

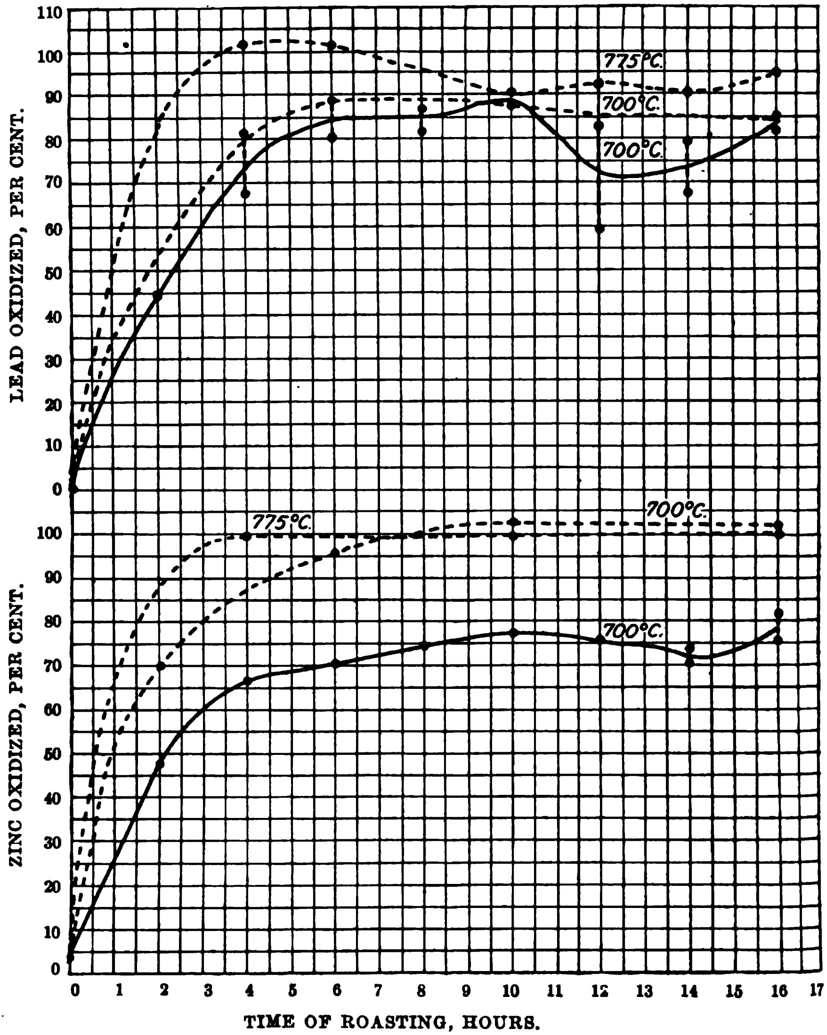

FIGURE 3.—Curves showing effects of leaching with different solvents, Daly-Judge No. 1 sample, 80-mesh size. Zinc: Solid-line curve, sulphuric acid solution; dotted curves, acetic acid solution. Lead: Solid-line curve, acidified ammonium acetate solution; dotted curves, acidified brine.

that can safely be used, in order to avoid the formation of ferrites. As much of the zinc is roasted to oxide and as ferrite forms on pro- ged roasting, it might be possible to obtain quick roasting at high peratures, provided the roast was discontinued promptly enough. is, of course, could apply only to ores low in iron.

LEACHING THE CALCINE.

As there was some doubt as to the relative value of sulphuric acid and acetic acid for recovering the zinc, and of acid ammonium acetate and acid brine for recovering the lead, comparative leaches were made on a calcine roasted at 700° C. of material ground to pass 80 mesh. The full-line curves in figure 3 give the recoveries with the acetic acid solvents, and the dotted lines, those for sulphuric acid and acid brine solvents, respectively. It can be seen that much more zinc is dissolved by the sulphuric acid solution than by the acetic acid solution, and that for practical purposes the leaches with acetic acid give low results. For this reason sulphuric acid is the leaching agent usually used in mills. In the same way a saturated brine, acidified with sulphuric acid, is a better solvent for the lead. Incidentally a roast at 775° C., the upper allowable limit of temperature, was made, with the result that practically all of the zinc was rendered soluble in sulphuric acid in less than four hours. Few of the commercial roasters will do such rapid work, and it would seem that roasters adapted to the use of very shallow beds of ore and to the use of finely divided ore might well be devised in order to obtain rapid roasting.

ROASTING OF A HIGH-IRON ORE FROM HONORINE MINE.

After roasting tests had been made with several sizes of this low-iron ore at different temperatures and for different lengths of time it was thought best to perform similar tests with an ore that was high in iron. Such an ore was available in the Honorine mine of the Bullion Coalition Mining Co., at Stockton, Utah; the sample obtained analyzed as follows: 13.1 per cent Zn, 10.56 per cent Pb, 22.2 per cent Fe, 30.32 per cent S, 18 per cent insoluble, and 2.42 per cent As.

The portions of this sample that were used for the tests to determine the effect of temperature and of length of roasting period gave the following screen analysis after preparation by crushing.

Results of screen analysis of ore from Honorine mine after crushing.

	Per cent.
On 20-mesh	2.6
On 35-mesh	24.1
On 65-mesh	22.4
On 100-mesh	10.6
On 150-mesh	7.8
On 200-mesh	5.5
Through 200-mesh	26.3

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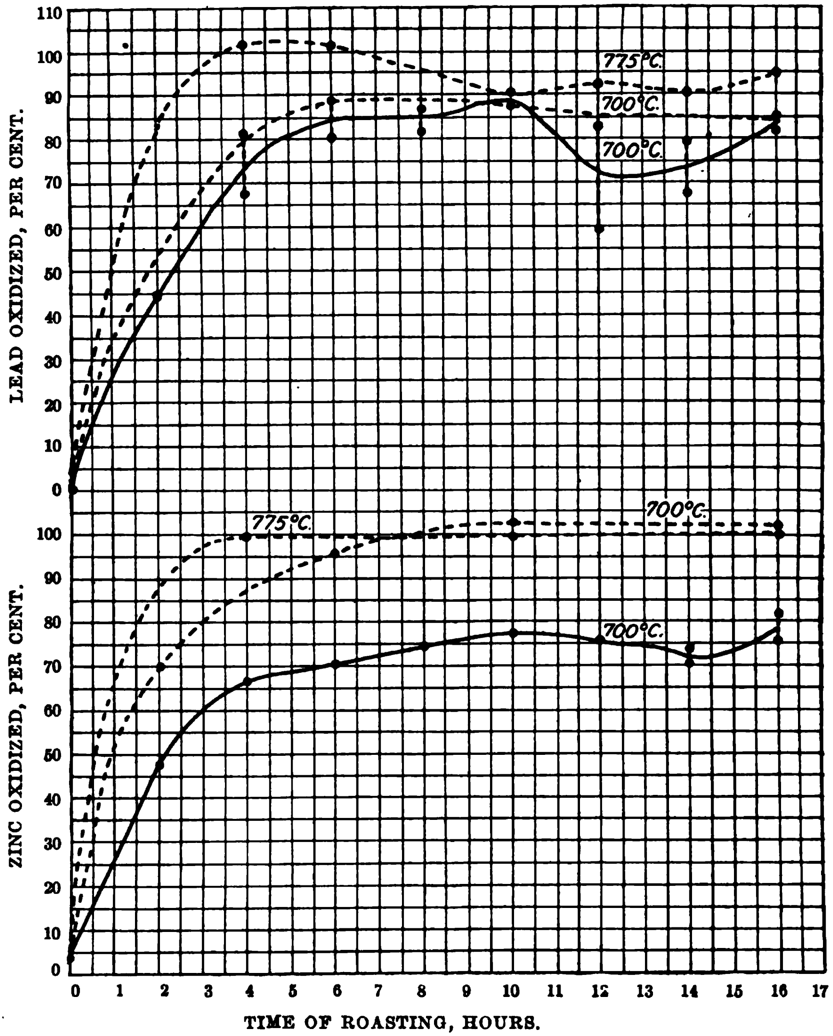

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Through 200-mesh	26.3

EFFECTS OF TEMPERATURE AND TIME OF ROASTING.

The lowest temperature investigated in this series of tests was 700° C., as the lower temperatures had been shown to be unsatisfactory for roasting zinc and lead minerals. Figure 4 shows that during the first two hours more soluble zinc is formed by roasting at 800° C. than by roasting at 700° C., owing to the more rapid roasting, but that after two hours the 800° roast contains less soluble zinc than the 700° roast, whereas the solubility of the zinc is very low after a 900° roast. This decrease in solubility was assumed to be due to the formation of ferrites, although the lead might be the cause of some of the trouble. However, as the lead did not cause any trouble in the first ore tested, this latter assumption is not regarded as probable. The

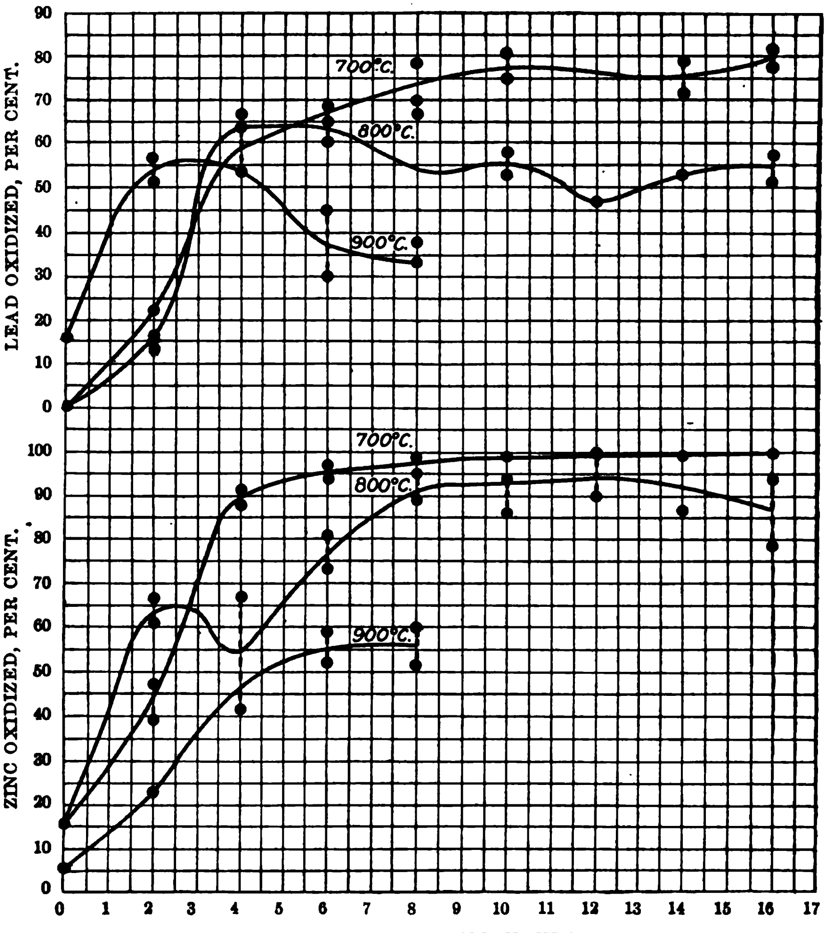

FIGURE 4.—Curves showing effects of temperature on speed of roasting; Bullion Coalition, No. 5 sample, 97.4 per cent passed 20-mesh.

curves show that it is not safe to heat the ore much above 700° C. during the roasting. Moreover, some of the lead is rendered insoluble on long roasting at high temperatures and the lead recovery curves show a decided drop with increasing time. Practically all of the zinc is rendered soluble in about six hours' roasting at 700°, whereas nearly 10 hours' roasting seemed to be necessary for maximum solubility of the lead.

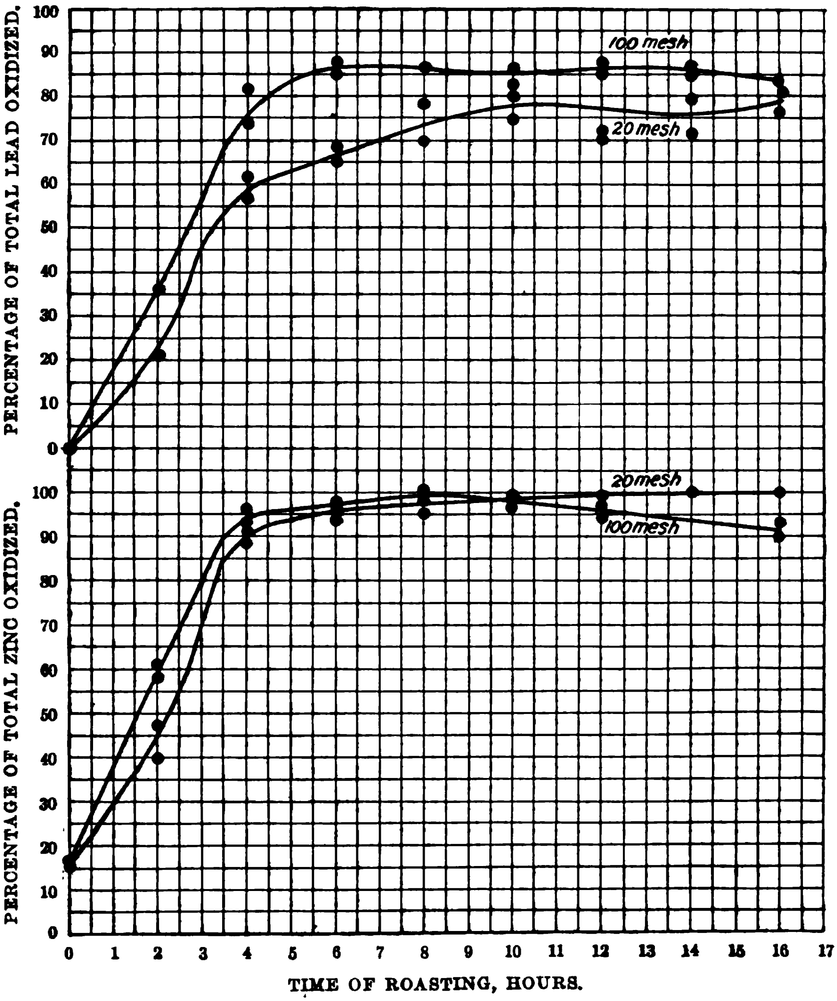

FIGURE 5.—Curves showing effects of size of ore on speed of roasting; Bullion Coalition, No. 5 sample, roasted at 700° C., 20-mesh and 100-mesh sizes.

EFFECTS OF FINENESS OF GRINDING.

Figure 5 shows the time-extraction curves for two roasts on material of different sizes, 20-mesh and 100-mesh. It is surprising how small a difference the size of the particles makes in the roasting of sphalerite. The material ground to pass a 100-mesh screen showed the gradual formation of ferrite during the last eight hours of the roast. This seems reasonable, as the ore is more finely ground and there is a better chance for the sphalerite being in contact with pyrite during the roasting, with the resultant combination of iron oxide with zinc oxide as time goes on. As regards the galena, the lead recoveries from the 100-mesh material were markedly higher, after a given length of time, than the corresponding ones for 20-mesh material. The maximum percentage of lead rendered soluble by roasting 20-mesh material was about 80 per cent, whereas nearly 90 per cent was rendered soluble in the roasting of the 100-mesh material.

EXPERIMENTS TO DETERMINE EFFECTS OF LEAD.

In these tests it was suspected that some of the lead after roasting tended to form lead silicate. To obtain light on this point, a finely ground slime from the dump of the Horn Silver mine at Frisco, Utah, was roasted. This material had the following analysis: 6.3 per cent Zn, 7.4 per cent Pb, 53.6 per cent insoluble, 4.4 per cent Fe, 2.2 per cent CaO, 6.13 per cent S, and 8.5 per cent Al_2O_3. The results of the two roasts made are plotted in figure 6. It can be seen that roasting at 650° C. renders practically all of the lead soluble in a very short time. Eighty per cent of the lead in this ore was oxidized before roasting and only 20 per cent of the zinc was oxidized. When the ore was roasted at 750° C., the solubility of the lead seemed to be less, although the recovery of zinc was much better, than from roasting at 650° C. However, the formation of ferrite of zinc at 750° C. seems to be quite plainly indicated by the fall in the last two-thirds of the zinc curve for that temperature.

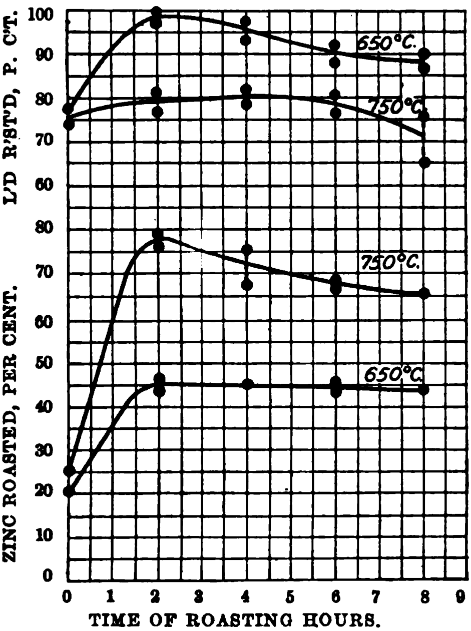

FIGURE 6.—Curves showing effects of temperature on solubility of zinc and lead in slime from Horn Silver dump, Frisco, Utah, 650 and 750° C.

Similar curves for the roasting of a series of ores all of which contained both lead and zinc, at 750° C., are shown in figure 7. The analyses of these ores are given in Table 15. Bullion Coalition No. 1 sample is representative of a large tailing dump from a concentration mill in which most of the zinc and lead sulphides had been removed from the pyritiferous gangue by gravity concentration. Bullion Coalition No. 2 sample is representative of a zinc-iron-lead middling product, made at this mill, which is run into the tailing. The American Flag No. 2 sample is representative of the partly oxidized ores of the Park City district of Utah, and the Daly-Judge No. 2 sample is representative of the deeper sulphide ores of that district when iron is present in larger amounts. The Ophir Hill No. 2 sample is of the slime tailing of the Ophir Hill mill at Ophir, Utah, not far from Stockton, where the Bullion Coalition mines are situated.

In general, the tests showed that for all of these ores the most favorable roasting temperature was 750° C. At this temperature the maximum proportion of soluble zinc obtained is formed in about four hours, if the roasting bed is not more than an inch deep. The same remark applies to the lead in these ores.

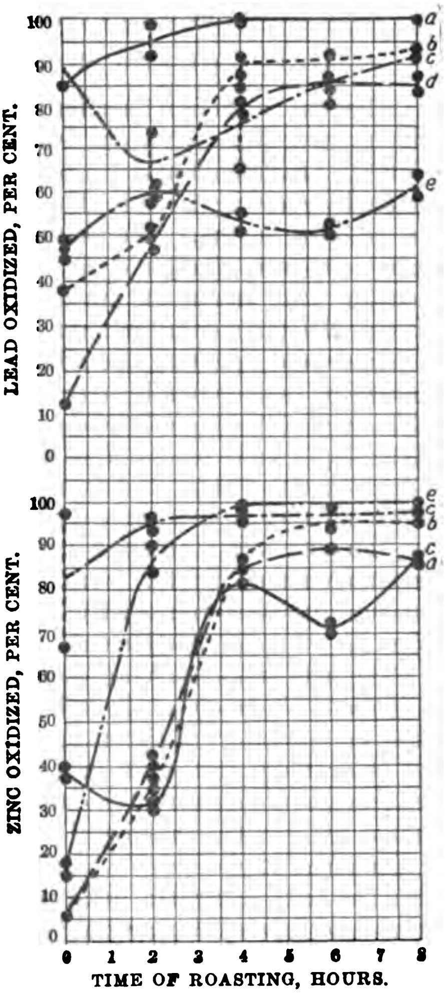

FIGURE 7.—Curves showing effects of roasting on solubility of zinc and lead in various ores, 750°C.: *a*, Bullion Coalition, No. 1 sample; *b*, Bullion Coalition, No. 2 sample; *c*, American Flag, No. 2 sample; *d*, Daly Judge, No. 2 sample; *e*, Ophir Hill, No. 2 sample.

TABLE 15.—*Results of analyses of zinc-lead sulphide ores.*

Mine and sample.	Insoluble.	CaO.	Fe.	Al_2O_3.	Zn.	Pb.	Cu.	S.	CO_2.	As.	Mn
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
Bullion Coalition, No. 1	10.5	2.1	33.0	7.6	4.4	3.0	0.10	37.34	3.53	1.13	......
Bullion Coalition, No. 2	9.3	3.4	28.8	15.8	8.5	10.5	.07	26.86	......	2.33	......
American Flag, No. 2	73.9	4.7	1.1	4.4	6.9	4.2	.11	.40	5.24	......	0.37
Daly-Judge, No. 2	22.0	5.1	16.4	1.9	12.0	15.4	.07	24.75	6.48	.20	......
Ophir Hill, No. 2	51.5	5.0	4.3	....	3.9	3.8	1.16	4.69	......	.25	......

LEACHING THE CALCINED ORE.

GENERAL METHOD USED IN TESTS.

After the best conditions for roasting the zinc in the ores had been determined the next step was to run larger leaching tests. As 750° C. had been found the best temperature, the experiments were made by roasting the ores at that heat and then leaching the calcines in 2.5-liter bottles. The calcines, before leaching, were ground to pass a 60-mesh sieve, in order that all lumps might be broken. Five leaches were made on the calcines of each roast, as follows: (1) Using the theoretical amount of sulphuric acid needed to dissolve the zinc and agitating the mixture for 1.5 hours; (2) using a slight excess of sulphuric acid over the theoretical amount and agitating 1.5 hours; (3) using a large excess of acid and agitating for 1.5 hours; (4) using a small excess of acid and leaching for 3 hours; (5) using a large excess of acid and leaching for 3 hours. This series was to determine whether an excess of acid was necessary for extracting all of the zinc and whether 3 hours' agitation would give any better extraction than did 1.5 hours. Previous work in the laboratory had shown that as a rule most of the zinc went into solution during the first few minutes of agitation.

The acid efficiency was calculated from the analyses of the solutions and the assay of the tailing. The percentage of zinc recovery was calculated from either the analysis of the solution or the assay of the tailing, and often as a check on the work from both. The recoveries as calculated by these two methods should serve as a check without the running of duplicate analyses.

In many of the tests the tailing from the acid leaches was again leached with saturated brine in order to remove the lead sulphate that had formed during the roasting or during the leaching of the zinc. Thus it was possible to determine whether an ore of mixed lead and zinc sulphides might be roasted, and the zinc and the lead leached separately.

TESTS OF ORE FROM DALY-JUDGE MINE.

The first ore treated was the Daly-Judge No. 1, previously described. The results of the leaching tests of this ore are given in Table 16. The first five tests seemed to indicate a slight advantage in treating the ore for 3 hours instead of for 1½ hours, but this is not true of the next five tests. In any event the difference is slight. The ore used up all of the acid when only the amount theoretically required to dissolve the zinc was added. Of course, some of the zinc sulphide had been roasted to sulphate, but the amount was small as determined a number of times by water leaches of the calcines. The temperature was high enough and the time of roasting long enough

TABLE 16.—*Results of leaching Daly Judge calcines.*

[Calcines from Daily Judge, No. 1, roasted at 750° C.]

Leach, No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Percentage of zinc in heads.	Zinc, grams.			Zinc (per cent) recovered.		Iron, grams.		Acid efficiency	Pulp ratio.	Lead in heads.		Quantity of brine used for leading lead, c. c.	Time of leaching lead, hours	Lead, grams.		Lead (per cent) recovered.	
		Per cent.	Grams.				In heads.	In tails.	In solution.	By tails assay.	By solution.	In heads.	In solution.			Per cent.	Grams.			In tails.	In solution.	By tails assay.	By solution.
1	1½	10	8.60	0.00	35	16.55	5.79	0.98	4.96	83.1	85.7	1.05	0.040	87.3	2	......	......	......	......	......	......	......	......
2	1½	10	11.64	1.12	35	16.55	5.79	.868	5.16	85.1	89.2	1.05	.047	74.2	3	......	......	......	......	......	......	......	......
3	1½	10	20.31	9.84	35	16.55	5.79	.812	5.21	86.2	90.1	1.05	.065	75.2	5	......	......	......	......	......	......	......	......
4	3	10	11.64	.616	35	16.55	5.79	.729	4.76	87.3	82.3	1.05	.075	65.1	3	......	......	......	......	......	......	......	......
5	3	10	20.31	9.10	35	16.55	5.79	.648	5.25	88.8	90.7	1.05	.080	70.7	5	......	......	......	......	......	......	......	......
1-a	1½	10	8.75	.00	35	16.55	5.79	1.29	4.62	77.8	79.8	1.05	.056	79.8	2	13.15	4.61	500	2	1.19	3.68	74.2	79.8
2-a	1½	10	11.75	1.494	35	16.55	5.79	.99	4.84	82.8	83.6	1.05	.095	69.9	3	13.15	4.61	500	2	1.08	3.80	76.6	82.5
3-a	1½	10	20.00	9.10	35	16.55	5.79	.82	4.92	85.7	84.9	1.05	.075	63.8	5	13.15	4.61	500	2	.91	4.025	80.2	87.3
4-a	3	10	11.75	1.122	35	16.55	5.79	1.12	4.84	80.6	83.6	1.05	.075	63.8	3	13.15	4.61	500	2	.91	3.94	80.2	85.5
5-a	3	10	20.00	8.86	35	16.55	5.79	.91	4.88	84.3	84.3	1.05	.079	62.0	5	13.15	4.61	500	2	.85	4.13	81.6	89.6

TABLE 17.—*Results of leaching calcines from the roasting of Bullion Coalition No. 5 ore.*

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Percentage of zinc in heads.	Zinc (grams) in—			Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.	Lead in heads.		Brine, c. c.	Time of leaching lead, hours.	Lead (grams) in—		Lead (per cent) recovered by—	
		Per cent.	Grams.				Heads.	Tails.	Solution.	Tails assay.	Solution.	Heads.	Solution.			Per cent.	Grams.			Tails.	Solution.	Tails assay.	Solution.
ROASTED AT 750° C. FOR 4 HOURS ON BOTTOM OF FURNACE.																							
1	1½	10	8.11	1.04	35	13.47	4.71	1.33	3.71	71.7	78.8	9.50	0.084	79.2	2	9.6	3.36	350	2	0.45	2.83	86.6	84.2
2	1½	10	11.11	4.25	35	13.47	4.71	.96	3.71	79.5	78.8	9.50	.103	81.6	3	9.6	3.36	350	2	.53	2.82	84.2	83.9
3	1½	10	20.00	13.20	35	13.47	4.71	1.17	3.64	75.0	77.3	9.50	.089	80.8	5	9.6	3.36	350	2	.59	2.78	82.4	82.8
4	3	10	11.11	4.09	35	13.47	4.71	1.09	3.72	76.8	78.9	9.50	.101	80.0	3	9.6	3.36	350	2	.52	2.86	84.5	85.1
5	3	10	20.00	12.92	35	13.47	4.71	1.13	3.66	75.9	77.7	9.50	.103	78.2	5	9.6	3.36	350	2	.56	2.78	83.4	82.8
ROASTED AT 750° C. FOR 4 HOURS IN CLAY DISHES.																							
1-a	1½	10	7.64	1.00	35	14.5	5.08	2.03	3.30	60.0	65.0	9.38	0.0077	75.1	2	9.38	3.28	350	2	1.42	1.75	56.7	53.8
2-a	1½	10	10.87	4.45	35	14.5	5.08	1.99	3.33	60.8	65.5	9.38	.0913	78.4	3	9.38	3.28	350	2	1.36	1.79	58.5	54.6
3-a	1½	10	20.00	13.25	35	14.5	5.08	1.97	3.41	61.2	65.8	9.38	.0982	76.3	5	9.38	3.28	350	2	1.35	1.83	58.8	55.8
4-a	3	10	10.87	3.94	35	14.5	5.08	1.86	3.41	63.4	65.8	9.38	.1150	74.4	3	9.38	3.28	350	2	1.33	1.80	59.4	55.0
5-a	3	10	20.00	13.05	35	14.5	5.08	1.81	3.43	64.3	65.8	9.38	.1265	74.5	5	9.38	3.28	350	2	1.30	1.86	60.3	56.7
ROASTED AT 700° C. FOR 6 HOURS IN CLAY DISHES.																							
1-b	1½	10	7.85	1.44	35	14.85	5.20	1.48	4.04	71.6	77.8	9.17	0.075	95.1	2	9.34	3.26	350	2	1.00	2.13	69.4	65.3
1-b	1½	10	10.87	4.53	35	14.85	5.20	1.38	4.06	73.5	78.2	9.17	.084	96.5	3	9.34	3.26	350	2	.87	2.23	73.2	68.4
1-b	1½	10	20.00	13.66	35	14.85	5.20	1.41	4.14	73.0	79.7	9.17	.089	98.6	5	9.34	3.26	350	2	.92	2.11	71.8	64.7
1-b	3	10	10.87	4.52	35	14.85	5.20	1.43	4.08	72.6	78.6	9.17	.083	97.1	3	9.34	3.26	350	(a)	1.05	1.85	67.8	56.8
1-b	3	10	20.00	13.58	35	14.85	5.20	0.93	4.20	84.1	80.8	9.17	.094	98.7	5	9.34	3.26	350	2	.77	2.51	76.4	77.0

[a] Not agitated.

TABLE 18.—*Results of leaching calcines from chloridizing roast of Bullion Coalition No. 5 ore.*

[Roasted at 750° C. for 4 hours in clay dishes with 10 per cent salt.]

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Percentage of zinc in heads.	Zinc (grams) in—			Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.	Lead in heads.		Brine, c. c.	Time of leaching lead, hours.	Lead (grams) in—		Lead (per cent) recovered by—	
		Per cent.	Grams.				Heads.	Tails.	Solution.	Tails assay.	Solution.	Heads.	Solution.			Per cent.	Grams.			Tails.	Solution.	Tails assay.	Solution.
1-c	1½	10	8.04	5.78	35	12.43	4.35	3.05	1.04	29.9	23.9	8.82	0.0685	69.5	2	6.06	2.12	350	2	0.102	2.00	94.9	94.3
2-c	1½	10	10.87	8.90	35	12.43	4.35	3.05	1.04	29.9	23.9	8.82	.0780	79.7	3	6.06	2.12	350	2	.077	2.03	96.2	95.8
3-c	1½	10	20.00	17.80	35	12.43	4.35	2.86	1.02	34.2	23.4	8.82	.0755	70.0	5	6.06	2.12	350	2	.077	2.01	96.2	94.9
4-c	3	10	10.87	8.90	35	12.43	4.35	2.66	1.08	38.8	24.8	8.82	.0880	82.7	3	6.06	2.12	350	2	.082	2.00	96.2	94.3
5-c	3	10	20.00	18.30	35	12.43	4.35	3.10	1.10	28.7	25.3	8.82	.0905	97.7	5	6.06	2.12	350	2	.095	2.03	95.6	95.8

so that usually most of the zinc sulphate formed was broken up. With this ore the recovery of zinc possible by the method of leaching used is apparently about 85 per cent, and the acid efficiency varies from about 65 to 75 per cent. About 80 per cent of the lead can be recovered from the tailing by leaching with saturated brine for two hours.

TESTS OF ORE FROM BULLION COALITION MINE.

The Daly-Judge ore was roasted on clay roasting dishes. In testing the Bullion Coalition No. 5 ore, roasting both in dishes and on the bottom of the muffle was tried. The results of these two tests, presented in Table 17, show that roasting on the bottom of the muffle was fairly successful, whereas roasting in dishes gave low recoveries, both of the zinc and of the lead. As it was thought that the temperature of roasting had been too high, another roast was made (see Table 17) at 700° C. and for a longer period (6 hours). The zinc recovery was better, but the lead recovery was somewhat low. This ore is difficult to roast on account of the large content of pyrite; seemingly the best recovery possible is about 75 per cent of the zinc and about 80 per cent of the lead, with about 80 per cent acid efficiency. For removing zinc from the calcines a 3-hour leach seems to show little improvement over a 1½-hour leach.

EFFECTS OF ROASTING WITH SODIUM CHLORIDE.

As some success was had in the removal of lead from such mixed sulphide ores by roasting with sodium chloride in tests reported under the subject of "Lead in complex sulphides," one roast was made in this series of tests in which 10 per cent of common salt was added to the ore. As a result of adding the salt (see Table 18), only a small proportion of the zinc sulphide was converted to soluble form, but the lead sulphide was rendered almost completely soluble. This result is much like that obtained in the work on lead, except that with a blast roaster the lead can be almost entirely converted to soluble form in the presence of zinc sulphide, whereas usually only 4 to 10 per cent of the zinc sulphide is converted. On that account salt is evidently not a desirable addition product in reverberatory roasting of sphalerite.

TREATMENT OF LOW-LEAD MILL PRODUCTS.

Next a series of roasts were made on mill products that did not contain lead in important amounts. Two samples of zinc concentrates of relatively high grade, one from Joplin, Mo., and the other from Midvale, Utah, were roasted and leached, to determine what recoveries of zinc might be expected under the conditions in the laboratory roaster. Also, a sample of zinc-iron middling from the wet-concentration mill at Midvale and a sample of ore from the Wasa mine at Hall, Mont., were treated. The Wasa mine ore contains most of the zinc as marmetite, the double sulphide of zinc and iron, in which iron sulphide is thought to be present in solid solution in the zinc sulphide.

The analyses of these four products is given in Table 19, and the screen analyses of the products that were roasted are presented in Table 20.

TABLE 19.—*Results of analyses of four zinc products low in lead.*

Ore.	Insoluble.	CaO.	Fe.	Zn.	Pb.	Cu.	S.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Joplin blende concentrates				66.2			
Midvale blende concentrates			5.33	55.6	1.86	1.16	26.95
Midvale zinc-iron middlings			24.87	28.25	3.66	1.00	26.66
Wasa mine ore	37.5	1.37	20.30	15.1	0.71	1.60	21.95

TABLE 20.—*Results of screen analyses of zinc ores and concentrates low in lead.*

Size of mesh.		Missouri zinc blende.	Midvale zinc blende.	Midvale zinc-iron middling.	Wasa mine ore.
On 20-mesh	per cent	7.8			
On 35-mesh	do	32.2			18.4
On 65-mesh	do	27.8	2.8	7.2	27.8
On 100-mesh	do	11.8	23.2	24.2	11.4
On 150-mesh	do	4.4	20.8	18.8	4.6
On 200-mesh	do	7.2	33.2	23.2	9.0
Through 200-mesh	do	8.6	18.6	25.8	28.6

The high-grade concentrate from Joplin, Mo., was almost pure blende of a beautiful orange hue, and any failure to recover all of the zinc could only be due to incomplete roasting. This test gave a measure of the success of the roasting tests that are recorded here. As can be seen from Table 21, roasting this material at 750° C. for five hours rendered only 89 per cent of the total zinc soluble in the leaching solutions. The conclusion is that the roast must have been incomplete. The tailing would have to be reroasted if this were the best work that could be done on a single roast. The work of nearly everyone who has roasted this type of material has shown that well above 95 per cent of the total zinc content can be rendered soluble by sufficiently lengthening the time of the roast, and on that account little apprehension need be had over any necessity of reroasting the residue. Also, in these tests an acid efficiency of more than 90 per cent was attained.

TABLE 21.—*Leaching zinc blende from Joplin, Mo., after roasting.*

[Roasted at 750° C. for 5 hours.]

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sam-sample, grams.	Percentage of zinc in heads.	Zinc (grams) in—		
		Per cent.	Grams.				Heads.	Tails.	Solution.
1	1½	10	26.18	2.2	25	66.2	16.56	2.02	14.3
2	1½	10	31.40	7.5	25	66.2	16.56	1.99	14.8
3	1½	10	41.80	17.0	25	66.2	16.56	1.90	14.42
4	3	10	31.40	7.4	25	66.2	16.56	2.01	14.80
5	1½	10	41.80	17.6	25	66.2	16.56	2.01	14.68

Leach No.	Time, hours.	Acid used.		Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.
		Per cent.	Grams.	Tails assay.	Solution.	Heads.	Solution.		
1	1½	10	26.18	87.2	86.4	0.5	0.035	90.1	9
2	1½	10	31.40	88.0	89.4	0.5	0.042	93.5	11
3	1½	10	41.80	88.5	87.2	0.5	0.035	87.8	15
4	3	10	31.40	87.8	89.4	0.5	0.044	93.1	11
5	1½	10	41.80	87.8	88.6	0.5	0.037	91.5	15

The roast of the Midvale blende concentrate, as seen in the first part of Table 22, was more successful, possibly because it was finer than the Missouri concentrate. Extractions of more than 96 per cent were obtained, and the acid efficiency was practically 100 per cent. Any lead present was rendered almost completely soluble in brine solution. The lack of agreement between the solution recoveries and the removal of lead as indicated by the analysis of the tailing indicate that about 25 per cent of the lead had volatilized during the roast. The success of this roast showed that with proper grinding the conditions of roasting chosen were designed to give high recoveries of the zinc and the lead.

In the treatment of the middlings from Midvale the recovery of zinc was not so high, possibly through the formation of ferrites during roasting. The results are given in the second part of Table 22. Practically all of the lead, which was present in small amount, was recovered by leaching in brine solution following the sulphuric-acid leaching for zinc.

TABLE 22.—*Results of leaching zinc-blende concentrate and zinc-iron middling from Midvale, Utah, after roasting.*

ZINC BLENDE CONCENTRATE.

[Roasted at 750° C. for 5 hours.]

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Per centage of zinc in heads.	Zinc (grams) in—			Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.	Lead in heads.		Brine, c. c.	Time of brine leaching, hours.	Lead (grams) in—		Lead (per cent) recovered by—	
		Per cent.	Grams.				Heads.	Tails.	Solution.	Tails assay.	Solution.	Heads.	Solution.			Per cent.	Grams.			Tails.	Solution.	Tails assay.	Solution.
6	1½	10	22.35	1.5	25	57.2	14.30	0.38	13.8	97.4	96.6	1.49	0.042	99.8	8	1.49	0.37	100	48	0.016	0.272	94.5	73.4
7	1½	10	26.18	5.6	25	57.2	14.30	.32	13.8	97.7	96.6	1.49	.035	101.2	9	1.49	.37	100	48	.014	.280	96.2	75.7
8	1½	10	41.80	20.6	25	57.2	14.30	.34	13.8	97.6	96.6	1.49	.044	98.3	15	1.49	.37	100	48	.008	.280	97.8	75.7
9	3	10	26.18	5.7	25	57.2	14.30	.36	13.8	97.5	96.6	1.49	.042	101.5	9	1.49	.37	100	48	.017	.252	95.4	68.2
10	1½	10	41.80	21.4	25	57.2	14.30	.30	13.92	97.9	97.4	1.49	.044	103.0	15	1.49	.37	100	48	.006	.280	98.8	75.7

ZINC-IRON MIDDLINGS.

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Per centage of zinc in heads.	Zinc (grams) in—			Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.	Lead in heads.		Brine, c. c.	Time of brine leaching, hours.	Lead (grams) in—		Lead (per cent) recovered by—	
		Per cent.	Grams.				Heads.	Tails.	Solution.	Tails assay.	Solution.	Heads.	Solution.			Per cent.	Grams.			Tails.	Solution.	Tails assay.	Solution.
11	1½	10	11.30	2.9	25	28.7	7.18	1.22	6.33	83.0	88.2	7.63	0.028	113.6	4.0	3.83	0.96	100	48	0.053	0.770	94.3	80.2
12	1½	10	15.64	7.1	25	28.7	7.18	1.54	6.46	78.6	90.0	7.63	.030	114.2	6.0	3.83	.96	100	48	.046	.784	95.2	81.6
13	1½	10	20.88	11.6	25	28.7	7.18	1.59	6.46	77.9	90.0	7.63	.033	105.0	7.5	3.83	.96	100	48	.007	.795	99.2	82.8
14	3	10	15.64	6.6	25	28.7	7.18	1.67	6.48	76.8	90.2	7.63	.030	108.3	6.0	3.83	.96	100	48	.026	.778	97.3	81.0
15	1½	10	20.88	13.3	25	28.7	7.18	1.59	6.46	77.9	90.0	7.63	.030	128.6	7.5	3.83	.96	100	48	.027	.832	97.2	86.6

TESTS OF MARMATITE ORE FROM WASA MINE.

Roasting and leaching of the marmatite ore from the Wasa mine gave low recoveries of the zinc; the maximum recovery possible seemed to be about 75 per cent. A 3-hour contact with excess of acid failed to dissolve any more zinc and only used up more acid, reducing the acid efficiency from 100 per cent, as indicated by a 1.5-hour leach, to 83 per cent. Low zinc recoveries are to be expected from such ore, on account of the intimate contact of iron and zinc in the mineral. Practically ideal conditions exist in this ore for the formation of zinc ferrite from the iron and zinc oxides formed in roasting, as the purest piece of blende contained about 12 per cent iron. A number of other roasts of this ore were tried, the results of which need not be recorded in this paper, but seemingly no better recoveries of the zinc could be obtained. The data for the first test are given in Table 23.

TABLE 23.—*Results of leaching ore from Wasa mine, after roasting at 750° C. for 4 hours.*

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Percentage of zinc in heads.	Zinc (grams) in—		
		Per cent.	Grams.				Heads.	Tails.	Solution.
1	1½	10	4.86	1.20	25	12.85	3.21	1.01	2.45
2	1½	10	7.48	3.81	25	12.85	3.21	1.00	2.47
3	1½	10	10.00	5.94	25	12.85	3.21	1.04	2.47
4	16	10	7.48	2.91	25	12.85	3.21	.93	2.48
5	3	10	10.00	5.61	25	12.85	3.21	.935	2.48

Leach No.	Time, hours.	Acid used.		Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.
		Per cent.	Grams.	Tails assay.	Solution.	Heads.	Solution.		
1	1½	10	4.86	68.6	76.3	5.24	0.0304	101.0	2
2	1½	10	7.48	68.8	76.9	5.24	.0421	101.5	3
3	1½	10	10.00	67.6	76.9	5.24	.0374	91.8	4
4	16	10	7.48	71.0	77.6	5.24	.0177	82.1	3
5	3	10	10.00	70.8	77.6	5.24	.0608	83.4	4

The conclusions are that with pure blende ground to pass about 60 mesh, practically all of the zinc can be roasted to a form soluble in sulphuric acid in two to five hours at 750° C., the time of roasting depending on the depth of ore bed used. Where zinc and iron are intimately combined, as in a marmatite ore, it is very hard to get all the zinc into a form soluble in acid, on account of ferrites of zinc forming. If the zinc and the iron in a mixed-sulphide ore are present as a mechanical mixture of pyrite and sphalerite, less ferrite is formed and recoveries of at least 90 per cent of the zinc are possible after a roast of the kind described. Most of these conclusions are not new, but there have not been much data on record to support them; the results presented here will make the truth more evident.

TESTS OF PARTLY OXIDIZED ORES.

As regards partly oxidized ores containing zinc sulphide, the question is whether such ores can be roasted in order to oxidize the sulphide of zinc, while the oxidized minerals of zinc remain unaffected, as far as their reaction toward sulphuric acid is concerned. A prominent example of such partly oxidized ore is that in the Lower Mammoth mine, of Mammoth, Utah.

ORE FROM LOWER MAMMOTH MINE.

Two samples of this ore were treated by roasting. Both of these were almost pure white, claylike material, much decomposed by weathering, and seemed to be completely oxidized, as far as the eye could distinguish. Such an ore would be almost impossible to leach if treated direct, on account of filtering difficulties, but would not be hard to leach after roasting, as the heat breaks up the "colloidal" material in the ore so that it can be filtered. The screen analyses of these two samples are given in Table 24. The results of the tests are shown in Tables 25 and 26.

TABLE 24.—*Results of screen analysis of two samples of partly oxidized ore from Lower Mammoth mine, Mammoth, Utah, after roasting.*

Size of mesh.		No. 1.	No. 2.
On 20-mesh	per cent..	15.8	....
On 35-mesh	do....	30.4	25
On 65-mesh	do....	45.0	68.4
On 100-mesh	do....	2.4	2.4
On 150-mesh	do....	1.8	0.6
On 200-mesh	do....	1.6	1.6
Through 200-mesh	do....	1.8	14

TABLE 25.—*Results of roasting and leaching partly oxidized zinc ore, Lower Mammoth No. 1.*

[Roasted at 750° C. for 4 hours.]

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Zinc (per cent) in heads.	Zinc (grams) in—			Zinc (per cent) recovered by—		Iron (grams) in—	
		Per cent.	Grams.				Heads.	Tails.	Solution.	Tails assay.	Solution.	Heads.	Solution.
1	1½	10	2.57	0.00	35	7.50	2.63	1.06	1.40	60.0	53.1	4.58	0.007
2	1½	10	5.00	.68	35	7.50	2.63	.86	1.53	67.0	58.0	4.58	.028
3	1½	10	20.00	15.32	35	7.50	2.63	.86	1.59	67.0	60.0	4.58	.035
4	2	10	5.00	.40	35	7.50	2.63	1.08	1.60	59.0	60.0	4.58	.023
5	3	10	20.00	15.24	35	7.50	2.63	1.10	1.60	58.0	60.0	4.58	.035

Leach No.	Time, hours.	Acid efficiency.	Pulp ratio.	Lead in heads.		Brine used in leaching for lead, c.c.	Time of leaching with brine, hours.	Lead (grams) in—		Lead (per cent) recovered by—	
				Per cent.	Grams.			Tails.	Solution.	Tails assay.	Solution.
1	1½	82.1	0.70	1.77	0.62	100	2	0.554	0.10	10.6	16.15
2	1½	51.2	1.25	1.77	.62	100	2	.411	.126	33.7	20.30
3	1½	51.3	5.00	1.77	.62	100	2	.340	.123	45.2	19.90
4	3	52.6	1.25	1.77	.62	100	2	.362	.123	41.7	19.90
5	3	50.9	5.00	1.77	.62	100	2	.345	.147	46.0	23.70

TABLE 26.—*Roasting and leaching partly oxidized zinc ore, Lower Mammoth No. 2.*

[Roasted at 750° C. for 4 hours.]

Leach No.	Time, hours.	Acid used.		Free acid, grams.	Weight of sample, grams.	Zinc (per cent) in heads.	Zinc (grams) in—		
		Per cent.	Grams.				Heads.	Tails.	Solution.
6	1½	10	5.20	0.00	35	11.45	4.00	1.13	2.75
7	1½	10	10.00	3.64	35	11.45	4.00	1.16	2.82
8	1½	10	20.00	13.68	35	11.45	4.00	1.15	2.93
9	3	10	10.00	3.44	35	11.45	4.00	1.11	2.92
10	3	10	20.00	13.16	35	11.45	4.00	1.01	2.89

Leach No.	Time, hours.	Zinc (per cent) recovered by—		Iron (grams) in—		Acid efficiency.	Pulp ratio.	Lead (per cent) in heads.
		Tails assay.	Solution.	Heads.	Solution.			
6	1½	72.0	68.8	1.77	0.009	79.8	1.3	0.00
7	1½	71.0	70.5	1.77	.023	67.0	2.5	.00
8	1½	71.3	73.2	1.77	.033	69.9	5.0	.00
9	3	72.3	73.0	1.77	.026	67.3	2.5	.00
10	3	74.7	72.3	1.77	.030	63.8	5.0	.00

With neither sample did it seem possible to roast the ore to advantage under the conditions that usually give good results in the treatment of zinc sulphide. The oxidized zinc minerals were so intimately associated with the oxidized iron minerals that the formation of ferrites in roasting was suspected; therefore a series of roasts were made to determine whether more careful roasting would take care of this difficulty. A quantity of ore was placed on a roasting dish and put into a cold muffle furnace. The temperature was gradually raised and the ore was rabbled intermittently. Grab samples of the calcines were taken at intervals, and these were treated with dilute sulphuric acid to determine the total amount of soluble zinc.

The effect of temperature on the solubility of the zinc is shown by the upper curve in figure 8, where the gradual temperature rise is plotted on one axis and the percentages of soluble zinc, as determined by the analysis of the solutions from the grab samples, on the other. At the same time, observations were made of the amounts of soluble iron. The sharp peak in the solubility curve for zinc illustrates how extremely sensitive this ore is to heat treatment. Heating the ore much above 700° C. seemed to cause the formation of insoluble zinc compounds, probably ferrites. The great amount of work, by other investigators, on the formation of zinc ferrites points to their being the cause of this decrease in solubility. All past experience has shown that in roasting such ore, zinc ferrites commence to form at a temperature slightly above 700° C. The increase in solubility up to that point is doubtless due to the desulphurization of the zinc sulphide present. This ore roasts rather slowly until a temperature above 600° C. is reached, and then, later, slight overheating above 700° C.

may spoil the calcine for leaching purposes. This is the most prominent instance of the formation of ferrites with which the writers are familiar.

The solubility curve for the iron is even more interesting. As is well known, pyrite becomes more soluble in acids on heating to a temperature hot enough to drive off some of the sulphur. Thus the rising solubility of the iron may be due either to this heating or to the partial desulphurization of the pyrite. However, the greater proportion of the iron in this ore is present as oxidized minerals, and these are rendered less soluble in dilute acids by heating. The downward trend of the curve at temperatures above 400° C. is

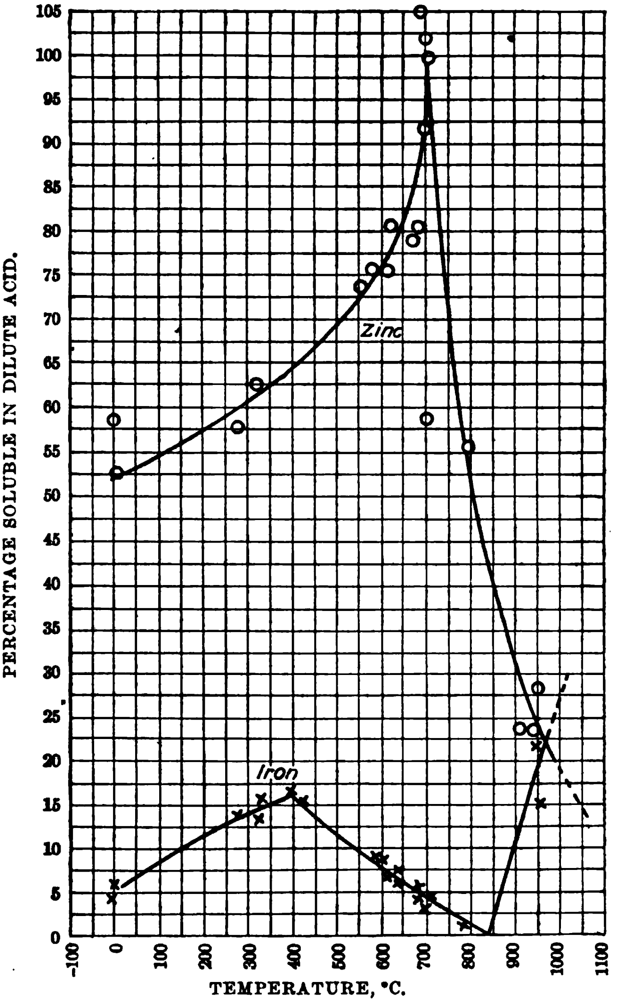

FIGURE 8.—Curves showing effects of roasting on solubility of zinc and iron in semioxidized zinc ore from Lower Mammoth mine.

TABLE 27.—*Results of roasting and sulphuric acid leaching, ore from Wasa mine.*

Test No.	Roasting period, hours.	Temperature, °C.	Loss of weight in roasting, per cent.	Time of leaching, hours.	Acid (H_2SO_4) added at start.		Free H_2SO_4 at finish, grams.	Weight of ore, grams.	Zinc (per cent) in—		Zinc (grams) in—			Zinc (per cent) recovered in—	
					Per cent.	Grams.			Heads.	Calcine.	Heads.	Tails.	Solution.	Solution.	Calcine.
1	2	750	14.2	1.5	8.9	16.89	8.98	35	12.9	14.5	5.075		4.11	81.0	24.2
2	3	750	14.4	1.5	8.9	16.89	8.90	35	12.9	14.5	5.075		4.02	79.2	24.2
3	4	750	14.6	1.5	8.9	16.89	8.90	35	12.9	14.5	5.075		4.03	79.5	24.3
4	5	750	15.1	1.5	8.9	16.89	8.90	35	12.9	14.5	5.075		3.99	78.6	34.4
5	7	750	15.3	1.5	8.9	16.89	8.90	35	12.9	14.5	5.075		3.92	77.3	24.5
6	2	730	13.6	1	8.9	14.61	7.61	35	12.9	14.95	5.230		3.79	72.4	24.2
7	3	730	14.6	1	8.9	14.61	6.70	35	12.9	15.10	5.280		4.16	78.8	24.4
8	4	730	15.3	1	8.9	14.61	7.42	35	12.9	15.20	5.320		4.03	76.0	24.4
9	2	920 730	13.6	1	8.9	11.25	4.33	35	12.9	14.95	5.230		4.03	77.2	24.2
10	3	730	14.6	1	8.9	11.25	3.49	35	12.9	15.10	5.280		4.11	77.8	24.4
11	4	730 920	15.3	1	8.9	11.25	4.21	35	12.9	15.20	5.320		5.24	98.5	24.4
12	2	850		1.5	11.5	11.5	5.81	35			4.65	2.94	1.90	40.9	
13	4			1.5	11.5	11.5	5.89	35		13.30	4.65	2.76	2.28	49.1	21.6
14	6	1,050		1.5	11.5	11.5	6.54	35		13.90	4.87	2.79	2.28	46.8	21.5
15	3	720	14.5	15	11.4	11.4	3.20	35		14.10	4.93	1.25	4.30	87.3	21.5
16	4	720	14.5	15	11.4	11.4	2.57	35		14.10	4.93	1.25	4.32	87.7	21.5
17	5	720	14.5	15	11.4	11.4	3.05	35		14.10	4.93	1.25	4.20	85.3	21.5
18	5	720	15.5	15	11.4	11.4	2.37	35		14.10	4.93	1.34	4.10	83.2	21.5
19	6	720	15.5	15	11.4	11.4	3.09	35		14.10	4.93	1.36	4.22	85.7	21.5

Test No.	Roasting period, hours.	Temperature, °C.	Loss of weight in roasting, per cent.	Iron (grams) in calcine.	Iron in solution.		Copper in calcine.		Copper in solution		Pulp ratio.	Acid efficiency, per cent.	Zinc in solution, per cent.	Remarks.
					Per cent.	Grams.	Per cent.	Grams.	Per cent.	Grams.				
1	2	750	14.2	8.47	0.070	0.105	0.23	0.08	0.04	0.061	4.3	78.7	2.7	
2	3	750	14.4	8.47	.073	.110	.23	.08	.04	.061	4.3	76.0	2.7	
3	4	750	14.6	8.50	.075	.112	.23	.08	.04	.061	4.3	76.5	2.7	
4	5	750	15.1	8.54	.081	.122	.24	.08	.04	.061	4.3	75.4	2.7	
5	7	750	15.3	8.57	.077	.115	.24	.08	.04	.061	4.3	74.3	2.6	
6	2	730	13.6	8.47	.077	.101	.23	.08	.05	.061	3.7	81.8	....	
7	3	730	14.6	8.55	.111	.111	.23	.08	.06	.055	2.9	79.5	....	
8	4	730	15.3	8.55	.123	.160	.24	.08	.04	.048	3.7	84.8	....	
9	2	920 730	13.6	8.47	.084	.084	.23	.08	.04	.037	2.9	114.1	....	Some lime in bath.
10	3	730	14.6	8.55	.074	.096	.23	.08	.04	.045	3.7	102.6	....	Do.
11	4	730 920	15.3	8.55	.075	.075	.24	.08	.03	.034	2.9	114.0	....	
12	2	850	....	....	.613	.613	.23	.08	....	....	2.86	50.5	1.9	
13	4	....	....	7.56	.227	.227	.23	.08	....	....	2.86	61.4	2.28	
14	6	1,050	....	7.52	.164	.164	.23	.08	....	....	2.86	69.4	2.28	
15	3	720	14.5	7.52	....	.033	....	....	....	....	2.86	79.2	4.3	
16	4	720	14.5	7.52	....	.033	....	....	....	....	2.86	73.8	4.32	
17	5	720	14.5	7.52	....	.0262	....	....	....	....	2.86	75.9	4.20	
18	5	720	15.5	7.52	....	.030	....	....	....	....	2.86	68.6	4.10	Roasted with MnO_2
19	6	720	15.5	7.52	....	.024	....	....	....	....	2.86	76.7	4.22	

doubtless due to this cause. The sharp up-turn beyond 850° C. was unexpected and the writers have been unable to explain it. However, as 850° C. is much above the permissible temperature for roasting zinc calcine, this latter part of the iron curve is of little practical importance.

MARMATITE ORE FROM WASA MINE.

Further tests were made with the marmatite ore from the Wasa mine, near Hall, Mont., by roasting and leaching with sulphuric acid. This ore, referred to in previous pages, contained a small amount of "rosin-jack," but the greater proportion of the zinc sulphide was black. The black zinc sulphide appeared homogeneous to the eye, even under the microscope, yet the purest crystal that could be selected contained 55 per cent Zn and 11.5 per cent Fe. Most of the crystals contained more than 16 per cent Fe. An analysis of this ore was as follows: Zinc, 12.9 per cent; iron, 20.72 per cent—insoluble, 38.10 per cent; lime (CaO), 1 per cent; copper, 0.20 per cent; and lead, 0.30 per cent. Another analysis has been given in Table 19 (p. 69).

The excess of iron is present chiefly as pyrite, with a small amount of pyrrhotite. As marmatite is supposed to be a solid solution of iron sulphide in zinc sulphide, the assumption may be made that roasting the zinc sulphide to the oxide without some zinc ferrite being formed along with the iron oxide would be a difficult matter.

The results of a series of roasts and leaches made in the laboratory at the Salt Lake City station by Walter Neal, representing McKeever Bros., under a cooperative agreement with the Bureau of Mines, are given in Table 27. For some reason the analysis of the solution for zinc gave unusually high titrations; hence, the recoveries, as calculated from the weight of zinc in the solutions, are disregarded. The tailing assays show that not more than about 75 per cent of the zinc could be recovered. However, this recovery was higher than was expected from a marmatite ore. On the assumption that the formula of zinc ferrite is $ZnFe_2O_4$, every unit of iron is combined with 0.58 unit of zinc. The flotation tests of this ore show that the purest zinc concentrate that could be made by roasting the pyrite to a nonflotative form contained 49 per cent Zn and 20 per cent Fe. Hence there is enough iron in the average marmatite particles to account for 23 per cent of the total zinc in the ore, provided that all of the marmatite iron is converted into ferrite iron by roasting, and that the ferrite of zinc is insoluble in the dilute acid solutions used.

The highest recovery of zinc, under the best conditions of roasting and leaching, was 74.7 per cent, or 25.3 per cent of the total zinc was insoluble. This latter figure compares closely with the 23 per cent calculated as ferrite. Consequently, the writers feel justified in tating that when marmatite ore is roasted, most of the iron present in the marmatite tends to unite with the zinc to form ferrite of zinc. Fortunately in most blendes such high percentages of "combined" iron are rare. Marmatite, being a solid solution of iron sulphide in zinc sulphide, has variable proportions of iron, but even the purest pieces rarely contain as much as 10 per cent. However, as all "black blendes" will tend to form ferrites of iron during roasting, the advisability of trying to leach such ores is doubtful. The only argument in favor of leaching ores of this type would be that the maximum zinc loss would not, as a rule, be more than about 10 per cent, although it might be higher for an ore, like the Wasa, having an unusually high content of combined iron.

MAGNETIC ROASTING.

By magnetic roasting of zinc ores is meant the process of heating zinc-iron sulphide mixtures to a sufficient degree to partly break up the pyrite into pyrrhotite and elemental sulphur. The lower sulphides of iron are magnetic, and sphalerite is nonmagnetic, hence magnetic roasting puts the ore into such a condition that it is capable of magnetic separation, provided the particles of magnetic iron sulphide and nonmagnetic zinc sulphide are mechanically separable. The data on some laboratory experiments along this line made at the Salt Lake City station are given under "Dry concentration processes" (pp. 45 to 53).

HYDROMETALLURGY OF ZINC SULPHIDE ORES.[a]

Hydrometallurgical treatment of zinc sulphide ores has been proposed in many different forms during the past half century, but has in general failed to be adopted, on account of the peculiar chemical properties of zinc and the low value of the products that can be made. Zinc requires more chemical or electrical energy, in comparison with its value after recovery, than almost any other major metal on the market; and the high consumption of chemicals or of electric energy involved in its recovery has usually made hydrometallurgical processes of questionable value.

Zinc sulphide resists most ordinary solvents, not being dissolved with sufficient rapidity by any of the commercial acids, hence requires roasting before the majority of the proposed hydrometallurgical processes can be applied. Sulphide of zinc is one of the most refractory sulphides to roast, as is mentioned in the part of this bulletin on roasting (p. 53), and on that account the cost of zinc roasting is higher than that for any other major metal. The lowest reported costs of oxide roasting in zinc smelters are $1.50 per ton of green ore, and in the older smelters $4 was not an uncommon cost. Chlorine gas is the only commercial medium that will attack zinc sulphide in the cold, and on this fact are based many of the "chloride" processes later described, their principal value being that the zinc sulphide need not be roasted.

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[a] Experimenters: O. L. Larson, A. E. Gartside, W. J. Woolf, R. W. Johnson, C. E. Sims, M. J. Udy, O. E. Young, A. J. McChrystal, and J. F. Cullen.

SULPHATE PROCESSES.

In order to get the zinc of the ore into solution as zinc sulphate a number of alternative processes are possible. The ore may be given a sulphating roast, or it may be given an oxidizing roast and then leached with sulphuric acid, or it can be mixed with strong sulphuric acid, without previous roasting, and heated in a reverberatory roaster until SO_3 fumes are evolved. This latter method has been tested in the laboratories at Anaconda and Great Falls, Mont.

A sulphating roast is usually rather difficult of accomplishment and the cost is high. It is reported that C. A. Hansen at Bully Hill, Cal., was able to sulphate as much as 90 per cent of the zinc in the ore from the Bully Hill mine. At Anaconda not more than 50 per cent is sulphated, and with many ores it is not possible to sulphate even 25 per cent of the zinc during roasting. Each ore seems to act differently and the free milling "rosin jack" ores do not sulphate as well as do the "black jack" ores. On the other hand, it is just as hard to roast the sulphide completely to zinc oxide, leaving no zinc sulphate in the ore, even by heating as highly as 1000° C. When a sulphate solution is to be made, the most desirable roast may be that which will either drive off all the sulphur or leave it as zinc sulphate, unless the consumption of sulphuric acid in a process is so large that the maximum sulphating effect is desired more than the formation of the maximum amount of soluble zinc.

Roasting of zinc sulphide ores, followed by leaching with solutions of sulphuric acid, has already been considered (pp. 53 to 79). In general the roasting should be so conducted as to convert all the zinc sulphide to soluble forms, yet the ore must not be heated above 700° C. on account of the tendency for the oxides of iron and of zinc to unite as ferrites of zinc. This makes the roasting a slow process; in spite of all precautions it is impossible with some ores to obtain good recoveries of the zinc by leaching. Thus the leaching plants at Anaconda, Bully Hill, and others have often been unable to get recoveries higher than 60 to 70 per cent. For a time the low recoveries with the sulphuric acid leaching process seemed to threaten its life, but supplementation of the process by igneous concentration of the zinc in the residue, as at Anaconda, seems to give it a better chance of survival.

If the gangue of the ore contains any acid-consuming substances, such as calcium carbonate or calcium oxide, or magnesia compounds or aluminum compounds soluble in sulphuric acid, the efficiency of the acid for dissolving zinc falls off. As enough acid must be used to supply the requirements of these compounds in addition to that of the zinc minerals, it does not take much of these compounds to make the consumption of acid excessive. Even with 100 per cent efficiency of the acid, 1.5 pounds are required to dissolve one pound of zinc. Therefore, if a zinc sulphide ore does not have a large content of acid-soluble gangue minerals, and roasting will leave no unoxidized blende, insoluble ferrites, basic sulphates, or other compounds, in the calcine, roasting and sulphuric acid leaching to recover the zinc is permissible. If the ore contains any considerable proportion of acid-consuming minerals any hydrometallurgical process of this kind will have to be preceded by mechanical concentration of the ore in order to remove such impurities, together with some of the iron sulphides that are not desirable.

SULPHITE PROCESSES.

A number of processes have been proposed involving the leaching of zinc calcines with a solution of sulphurous acid to form zinc bisulphite, which is sufficiently soluble in water to make such a method a commercial possibility. The original methods involved precipitation of the zinc as monosulphite of zinc, which is insoluble, by warming the solution of bisulphite. In heating the solution some sulphur dioxide would be liberated, and this could be caught and used again with that obtained from thermal decomposition of the zinc monosulphite in a furnace heated to a sufficiently high temperature.

DIFFICULTIES OF APPLYING PROCESS.

In practice some difficulties in the application of this process were met. In roasting zinc sulphide ores it is possible to obtain roaster gases containing about 5 per cent SO_2. The strength of the solution that can be prepared from this gas varies from 0.5 to 1.5 per cent SO_2. On the assumption that the solution will average 1 per cent SO_2 and that the ore to be treated contains 30 per cent zinc, 60 tons of this 1 per cent solution will be required to leach the zinc completely from 1 ton of ore. This solution ratio is too large for economical work in any hydrometallurgical process. Consequently the roaster gas would have to be applied directly to a pulp of the ore, rather than being dissolved by water in an absorption tower. Pulp can not be treated in a tower as it would soon choke the openings, hence some type of mechanical agitator for applying the gas to the pulp would be necessary. As far as the authors can learn, no such machine has ever been tried. The solubility of zinc bisulphite is such that about 6 per cent zinc can be easily obtained in solution, hence a 30 per cent ore would require 5 tons of solution per ton of ore, a volume which is very easily handled in the machines now available. Possibly an apparatus comprising a rotating drum, in which the gases and the pulp would pass in countercurrent, would be acceptable for this work, the zinc oxide of the ore combining with the dissolved sulphur dioxide fairly rapidly, so that more sulphur dioxide could dissolve in the water until all the zinc oxide had been converted to monosulphite of zinc, after which the addition of sulphur dioxide would form bisulphite of zinc until all the zinc had gone into solution.

In order to design such a machine properly, the rate of reaction of sulphur dioxide solutions of various strengths on roasted zinc sulphide ore, and the rate of absorption of sulphur dioxide gas by water in a tumbling barrel of this type would have to be determined. It is known that the rate of absorption of gases in liquids, when the liquids are showered down through the gases, is extremely rapid and depends on the effectiveness with which the water is sprayed. Nothing was known as to the rate of reaction between the sulphurous acid solutions and the ordinary zinc oxide calcine.

EXPERIMENTS.

Therefore, a series of tests were made on such a calcine with different strengths of sulphur dioxide solution for different time intervals, in order to obtain an idea of the reaction velocity. A sample of the zinc sulphide concentrate from the Daly-Judge concentrating mill, at Park City, Utah, was roasted and the calcine was treated with measured quantities of sulphurous acid solutions of known strength. Treatment was in 2.5-liter bottles placed on their sides on a rolling agitator to give much the same motion as that given to pulp in a tube mill or similar rotating barrel arrangement. The sample analyzed 46.6 per cent Zn, 7.4 per cent Fe, 1.86 per cent Pb, 1.54 per cent Cu, and 30.44 per cent S. The calcine contained 49.6 per cent Zn. The screen analysis of the calcine was as follows:

Screen analysis of calcine.

	Per cent.
On 20 mesh	0.05
On 35 mesh	17.0
On 65 mesh	47.05
On 100 mesh	17.80
On 150 mesh	6.85
On 200 mesh	6.05
Through 200 mesh	5.25

In the calcine 10.8 per cent of the zinc was present as zinc sulphate. The results of the tests are plotted in figure 9. In each test the theoretical amount of sulphur dioxide needed to render all the zinc soluble was used. Consequently, as 10.8 per cent of the zinc was present in the calcine as sulphate, about 10 per cent excess SO_2 over that necessary to leach the remainder of the zinc was present. From the curves it can be seen that even with 0.2 per cent solutions of SO_2, all the soluble zinc was dissolved in about 30 minutes. Only 96 per cent of the total zinc was soluble, the other 4 per cent presumably being present as unaltered sulphide or as ferrite. In a

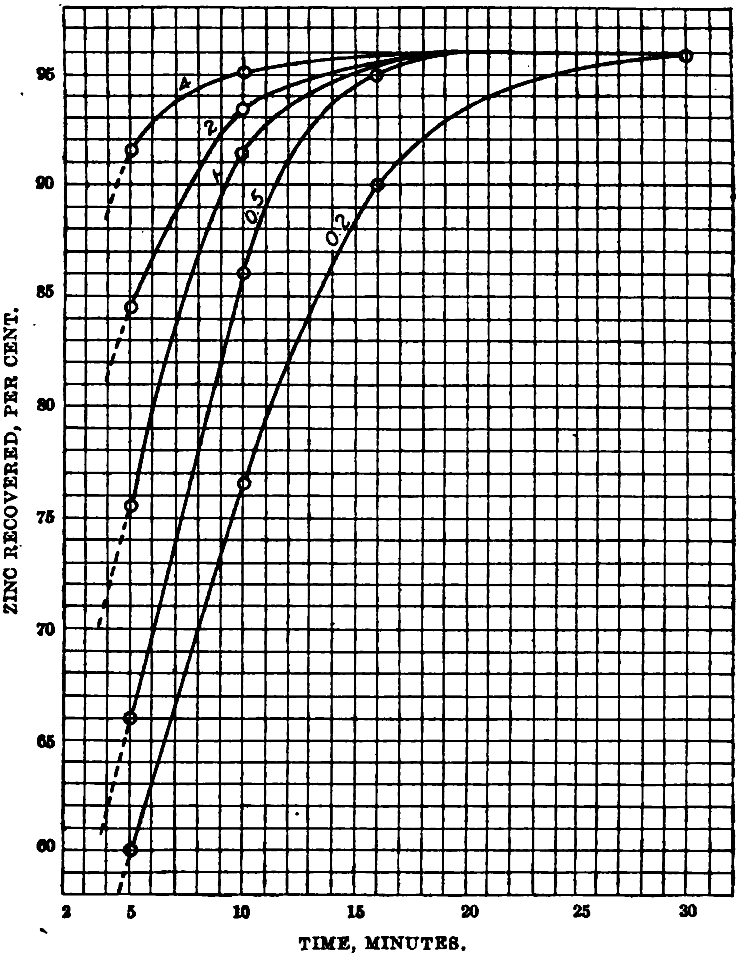

FIGURE 9.—Curves showing effect of strength of sulphurous acid on velocity of solution of zinc oxide. Figures near each curve indicates percentage of SO_2 in solution used.

rotating barrel, with continuous feed and discharge through the trunnions, and countercurrent application of the roaster gases, the tendency would be for the strongest solution of sulphur dioxide to form at the point where the pulp is discharged, the pulp traversing the barrel in solution of gradually increasing strength. With a 1 per cent solution, 91.5 per cent of the zinc is dissolved in 10 minutes and all of the soluble zinc in 15 minutes. Hence it is probable that the pulp would not need to remain in the tumbling barrel more than 15 or 20 minutes, provided that enough roaster gas could be introduced into the barrel in that time.

These figures are sufficiently encouraging to warrant the testing of such a solution barrel on a semicommercial scale by those who are interested in the possibility of this method.

As is shown in a subsequent part of this report (p. 100) dealing with precipitation of solutions, recovery of the zinc from the bisulphite solution by boiling is difficult because zinc bisulphite readily oxidizes to zinc sulphate. If it is expected to recover strong sulphur dioxide vapor from this boiling, to be used in making stronger solutions for leaching, trouble will be experienced and the process will not be satisfactory. If the ore contains a large excess of sulphur to be roasted off and only the roaster gases are used for leaching, there should be no difficulty in providing enough sulphur to carry on the work. A further discussion of this subject will be found under the leaching of oxidized zinc ores (pp. 130 and 131).

In testing several sulphide ores by this process it was found that solutions of sulphurous acid gave the same recoveries as did the sulphuric-acid solutions, and that the same chemical reactions took place. If the solutions are allowed to become neutral in order that iron and other impurities may be precipitated by hydrolysis in the presence of limestone, the insoluble zinc monosulphite tends to form so that the ultimate recovery of zinc suffers. Hence it may be said that purification of sulphite solutions of zinc before filtering, is still an unsolved problem, and that all the impurities in the solution must be allowed to remain until after the residue has been filtered off.

LEACHING AGENTS.

HYDROCHLORIC ACID.

The use of hydrochloric acid as a solvent for the leaching of zinc sulphide ores might be considered in localities having industrial plants where the acid is a useless by-product. The possibility of making Le Blanc soda at Great Salt Lake might be considered if hydrochloric acid could be used for that purpose. The advantage of using Le Blanc soda would be in the preparation of a solution of zinc chloride from which zinc hydrate could be precipitated by the use of lime, thus permitting the preparation of a zinc-oxide product suitable for some purposes, by utilizing only cheap chemicals. The use of hydrochloric acid on zinc carbonate ores was tested at the Salt Lake City station with some success. (See section on "Oxidized Ores," pp. 121 to 129.) Chloridizing roasting with salt, followed by water leaching, failed to give satisfactory recovery of the zinc.

HYDRO-FLUOSILICIC ACID.

Hydro-fluosilicic acid was recommended as being a desirable solvent for roasted zinc ores, on account of the ease of precipitating zinc from such solutions by electrolytic methods. However, this acid proved to be no better solvent of the zinc oxide, resulting from roasting the zinc sulphide, than was sulphuric acid or sulphurous acid, and its use had no particular advantage, as far as leaching is concerned. Hence, experiments with this reagent were discontinued. Reference to the tests with carbonate ores of zinc (p. 131) will reveal a good reason for not carrying this work any farther; also, the cost of preparing the acid is another good argument against its use, because large amounts are consumed by gangue minerals in the ores. The acid is used in lead refineries, but the losses in such work are much smaller than those which would be encountered in leaching roasted zinc ores.

CAUSTIC SODA.

Caustic soda has been proposed as a solvent for zinc oxide in many different processes. In these, soluble zincates of sodium are formed, and the recovery of the zinc is usually by electrolysis. Caustic soda solutions might be used in the presence of large amounts of acid-consuming gangue, thus avoiding acid losses, but the solutions used must be so strong (20 to 30 per cent NaOH) that serious losses of caustic soda in the tailing will result unless the tailing is carefully washed and the dilute wash solution evaporated. As evaporation is a relatively expensive process, this procedure would not be advisable. Moreover, the zinc sulphate always formed in roasting will tend to form sodium sulphate, using up further caustic soda, and any other sulphates present, such as iron or aluminum sulphates will cause still further consumption of the solutions. The sodium sulphate would have to be removed periodically and for this purpose lime is not efficient. Barium hydrate would be a good reagent for this purpose but would be liable to prove too expensive.

AMMONIACAL SOLUTIONS.

Ammoniacal solutions for leaching roasted zinc sulphide ores offer a more promising method. Mixtures of ammonia and ammonium carbonate are advocated by Bretherton,[a] whose papers have been before the public for some years past. Such solutions dissolve zinc oxide, zinc sulphate, and zinc carbonate, but do not attack zinc sulphide, zinc ferrite, or zinc silicate. Consequently, in the treatment of roasted sulphide ores, only the same compounds as those dissolved by sulphuric acid are soluble in these solutions.

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[a] Bretherton, S. E., The treatment of complex ores by the ammonia-carbon dioxide process· Trans. Am. Inst. Min. Eng., vol. 49, August, 1914, p. 802.

As far as can be learned, the leaching seems to offer no difficulty, the principal problem being the precipitation of the zinc from solution and the recovery of the ammonia. To date the proper adaptation of ammonia-handling machinery to this work has not taken place.

LEACHING IRON FROM ZINC-IRON SULPHIDES.

On account of the difficulty of roasting zinc sulphide completely, the testing of a proposed process for separating mixtures of zinc and iron sulphides by removing the iron, in place of leaching the zinc, was deemed worth while. On heating pyrite in a reducing atmosphere pyrrhotite or iron matte is formed, depending on the temperture, and both of these react quickly with sulphuric acid, forming ferrous sulphate and hydrogen sulphide. Pyrrhotite is about 25 times as easily dissolved as sphalerite and 75 times more easily than pyrite. Hence, the proposed process involves heating a mixture of zinc and iron sulphides, which is a common type of complex sulphide ores, in a reducing atmosphere in order to render the iron sulphide minerals easily soluble in sulphuric acid. Such a process would be useful at smelters needing an outlet for the disposal of sulphuric acid and sulphur. The pyrite loses its "feeble atom" of sulphur during such treatment, the sulphur condensing from the reducing gas after cooling. The hydrogen sulphide from the reaction of sulphuric acid on the calcine can be easily absorbed by lime, making calcium sulphide, a valuable product, or any similar sulphide can be made. Possibly a market could also be found for the iron sulphate formed.

The conditions requisite for this process were determined with a combustion tube. It was found that temperatures between 600° and 950° C. were the most satisfactory; at temperatures outside of this range the recoveries of iron were lower. Coal gas from the city mains, carbon monoxide gas, and hydrogen sulphide gas were all found to be suitable reducing atmospheres. With carbon monoxide and coal gas, some of the sulphur distilled from the pyrite gave an ill-smelling compound, COS, occasionally masked by the smell of other organic compounds of sulphur. The presence of both steam and air was found to be fatal to the process, for these seemingly oxidized the iron to such a condition that it was superficially resistant to the dilute sulphuric acid. However, by preliminary heating in air, most of the "feeble atom" of sulphur could be removed; then a short subsequent treatment in a dry reducing atmosphere would convert the residue to a condition such that it would react with the dilute acid.

DESCRIPTION AND RESULTS OF TESTS.

A sample of the ore from the Greenwood mine, at Pioche, Nev., which is so intimately crystallized that no method of mechanical separation has proved of much benefit, was tested by this process. The average results were as follows:

Results of test of ore from Greenwood mine.

	Original.	Residue.	Recovered in solution.
Zinc (Zn) .... Per cent..	21.35	24.91	11.0
Iron (Fe) .... do....	16.13	3.38	84.1
Lead (Pb) .... do....	9.54	12.12	4.5(?)

The original ore contained too little combined zinc and lead to be shipped profitably to a pigment plant making leaded zinc oxide, but the residue contained enough to be acceptable.

Middling from the wet-concentrating mill at the Midvale smelter of the United States Smelting Co. was likewise tested with favorable results:

Results of test of ore from Midvale smelter.

	Original.	Residue.	Recovered in solution.
Zinc .... per cent..	28.25	41.1	2.6
Iron .... do....	24.87	10.7	71.2

Middling from the Mary Murphy mine in Colorado, the Big Four mill near Park City, Utah, and similar materials gave like results.

In all the ores containing only mixtures of pyrite and sphalerite no difficulty was met in removing the iron. However, in ores containing "black jack" or marmetite, the iron is known to be in solid solution in the zinc sulphide, and hence is not attacked by the dilute sulphuric acid. Examples of the average figures on two such materials follows, the first being middling from the Comet mine at Basin, Mont., and the other being ore from the Cleveland mine at Silver City, N. Mex.

Results of tests of materials from Comet mill and Cleveland mine.

	Original.	Residue.	Recovered in solution.
Middling from Comet mill:			
Zinc .... per cent..	21.1	25.0	13.9
Iron .... do....	25.12	18.9	54.7
Lead .... do....	7.72	9.5	....
Ore from Cleveland mine:			
Zinc .... per cent..	23.5	43.5	23.1
Iron .... do....	30.2	16.6	77.15

Usually zinc ore-buying contracts are so worded that some of the original materials mentioned can not be sold to any advantage by the average shipper in the intermountain region. Some of the products of this process, however, are well within the range of profitable work, and in spite of their loss in weight from loss of sulphur and iron, are worth more, in actual gross value, than the original material. With spelter at 5 cents per pound and the usual contracts, the increase in value of the material plus the saving from decreased cost of freight to the smelter of a zinc-iron sulphide mixture containing about 30 per cent Zn, amounts to between $10 and $14 per ton of original material.

The only important difficulty with the process is that it requires a dry reducing atmosphere at a temperature of 600° to 900° C., conditions which are difficult of technical attainment. Probably dry carbon monoxide gas could be provided at a reasonable figure by using dry coke in a gas producer, but to design a retort or similar furnace for utilizing this gas would be difficult. Any chance oxidation due to flue gases or to water vapor from combustion gases must be avoided. Probably after cooling and condensing the sulphur from the reducing gases issuing from the furnaces, the gases could be either used again or burned under the furnaces. On account of the difficulty in designing a satisfactory furnace of simple design that would meet all the conditions, the writers have temporarily abandoned further investigation of this process.

PRECIPITATION OF ZINC SOLUTIONS.[a]

The successful precipitation of solutions containing zinc is the main factor on which hinges the commercial possibility of any hydrometallurgical process for zinc ores. Such great purity is required of any zinc compounds usually marketed that chemical precipitation of such a product as zinc oxide, in a form commercially acceptable, is a difficult matter. Also, when electrolytic precipitation is to be employed, minute amounts of impurities are often fatal to good deposition.

ELECTROLYTIC DEPOSITION.

A separate report on electrolytic zinc is being prepared by the Bureau of Mines, and that phase of the subject will not be taken up in this bulletin.

CHEMICAL PRECIPITATION.

With some of the more unusual solvents that have been proposed, chemical precipitation of zinc oxide or similar marketable products from the solutions has been sought.

AMMONIACAL SOLUTIONS.

Precipitation of ammoniacal solutions of zinc by boiling drives off most of the ammonia and leaves a precipitate of zinc hydrate which can be calcined to zinc oxide. The process permits the preparation of reasonably pure zinc oxide, because so few impurities of the zinc ore are soluble in ammonia. Copper compounds can be precipitated from these solutions before boiling, by the use of metallic zinc. As hinted at in the discussion of leaching (p. 86), the chief difficulties are due to deterioration of the solution and to losses of ammonia in the tailing and by vaporization. Machinery capable of leaching ores in confined tanks, and of absorbing all the ammonia evaporated from the solutions at any part of such a plant would have to be invented. Further, the cost of heating and boiling solutions to recover ammonia is somewhat higher than any of the unit costs usually allowed in plants treating ores of this kind. Ammonia sulphates and chlorides and other compounds formed in the leaching of the ore would have to be decomposed, in order to liberate the ammonia from the solution, by the use of lime after the zinc had been precipitated, and this would add a complication. The method deserves further study, but at the present time it is probable that methods can be developed which will not call for such complicated plants.

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[a] Experimenters: O. E. Young, C. W. Frith, H. J. Morgan, and R. G. Hinman.

CAUSTIC ALKALI SOLUTIONS.

Caustic alkali solutions of zinc (sodium zincate) can be decomposed by dilution, zinc oxide being precipitated, but evaporation of the solution after precipitation, in order to obtain a strong solution of caustic for further work, is rather expensive. Such solutions can also be electrolyzed, but usually give a spongy deposit, and the market for zinc dust is limited. Consequently there seems to be nothing to favor processes involving the use of caustic alkali solutions.

SULPHATE SOLUTIONS.

Sulphate solutions of zinc might be precipitated by such reagents as ammonia or caustic soda, giving a hydrate of zinc and a sulphate of the ammonia or the alkali metal which could be recovered or discarded. As regards ammonia, the solution of ammonium sulphate would later have to be boiled with lime added, in order to recover the ammonia for reuse. Many of the same objections that apply to the direct leaching of ore with ammonia solutions also apply to this latter process. Furthermore, the amount of ammonia absorbed in the precipitate would be very liable to make the losses of ammonia too high. Caustic soda, as a precipitant, is too expensive. Even crude sodium carbonate can not be delivered to many points in the intermountain region or elsewhere for less than 1 cent per pound. Beside, the sulphuric acid used in leaching the ore is not regenerated as in electrolytic precipitation. This is not a serious drawback as sulphuric acid can be made very cheaply as soon as a market for it is assured.

If lime could be used as a precipitant it would be much cheaper than either ammoniacal or alkaline precipitants; also most precipitates of metal hydrates made with lime are well flocculated and easy to filter. On that account rather thorough tests were made of a method of using lime as a precipitant of zinc sulphate solutions, which involved converting the zinc sulphate solution to zinc chloride solution by the use of a solution of calcium chloride, followed by precipitation of the zinc as zinc hydrate by the use of lime. The proposed method involved running a solution of calcium chloride onto the ore, adding sulphuric acid to bring the zinc into solution, leaving calcium sulphate in the ore and yielding a solution of zinc chloride containing the excess of calcium chloride. In precipitation of this solution with lime the calcium chloride would be regenerated. The reactions involved are:

$$CaCl_2+H_2SO_4+ZnO \text{ (in ore)}=CaSO_4+\text{Residue}+H_2O+ZnCl_2$$
$$ZnCl_2+Ca(OH)_2=Zn(OH)_2+CaCl_2$$

This process utilizes two very cheap chemicals, sulphuric acid and lime. Enough calcium chloride would have to be added to make up for mechanical losses of that chemical. Consequently, investigation of the two reactions outlined above was necessary.

PRECIPITATION OF ZINC SULPHATE SOLUTIONS WITH CALCIUM CHLORIDE.

The first step was to see whether zinc sulphate solutions could be converted to chloride of zinc by the use of calcium chloride. To do this, a series of fractional precipitations of the sulphates were made by adding to known amounts of zinc sulphate solutions 0.5, 0.6, 0.7, 0.8, 0.9, 1, and 1.1 equivalents of calcium chloride. The results are contained in Table 28. In each test the calcium chloride solution acted on the zinc sulphate solution for about 60 minutes before filtration. No filtration difficulties in the handling of the calcium sulphate precipitate were experienced. The solution of zinc used contained 4.24 grams of zinc per 100 c. c.

TABLE 28.—*Precipitation of sulphates from zinc sulphate solutions with calcium chloride.*

$CaCl_2$ equivalent.	Zinc content of precipitate.	Zinc lost.	SO_4 precipitated.
	Per cent.	*Per cent.*	*Per cent.*
0.5	0.56	1.18	50.2
.6	.51	1.13	58.8
.7	.15	.41	65.7
.8	.05	.18	74.6
.9	.05	.18	81.7
1.0	Nil	Nil	87.2
1.1	Nil	Nil	93.0

From these tests it can be seen that while an 0.5 equivalent of calcium chloride will precipitate the calculated amount of calcium sulphate from a zinc sulphate solution containing 5 per cent Zn, the theoretical equivalent to all the sulphates will precipitate only 87 per cent of the sulphates. The remaining 13 per cent of the sulphates is probably dissolved calcium sulphate. There is no published figure on the solubility of calcium sulphate in zinc chloride solutions, but the solubility of $CaSO_4$ in hydrochloric acid solution of 6 per cent strength is 6.12 parts per 100 parts of water, whereas the solutions used contained 2.36 parts at 25° C. As the solutions contained nearly 5 per cent Zn, or more than 10 per cent $ZnCl_2$, the conclusion drawn is probably justified. One encouraging feature was that the precipitated calcium sulphate contained almost no zinc. When the theoretical or slightly more than the theoretical amount of calcium chloride was used no zinc was adsorbed in the precipitate and lost.

PRECIPITATION OF ZINC CHLORIDE SOLUTION WITH LIME.

As the tests had shown that fairly efficient precipitation of the sulphates from solution could be obtained by use of calcium chloride, tests on the precipitation of zinc chloride solutions with lime were next in order.

TESTS WITH PURE ZINC CHLORIDE.

At first pure zinc chloride solutions were prepared, in order that the analysis of the precipitate might be studied without having any complicating effect due to the presence of dissolved calcium sulphate. Two solutions of zinc chloride were prepared, one containing 10 per cent Zn and one 5 per cent. Each of these solutions was tested with a series of additions of lime in amounts varying from 0.5 to 1.0 equivalent to the zinc. The mixtures were allowed to stand for 24 hours to insure complete reaction. The lime used contained 47.4 per cent CaO and 29.9 per cent MgO. The data on these tests is given in Table 29.

TABLE 29.—*Precipitation of zinc chloride solutions with lime.*

Zinc content of solution.	CaO equivalent.	Precipitation.	Content in air-dry precipitate.		
			Zinc.	CaO.	MgO.
Per cent.		*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
5	0.5	51.0	57.7	1.50	2.03
5	.6	61.8	53.8	1.98	2.39
5	.7	71.6	53.6	1.95	2.22
5	.8	83.2	53.7	2.18	2.06
5	.9	92.3	51.5	2.76	3.51
5	1.0	99.5	51.6	3.06	4.30
10	.5	57.0	49.8	2.62	2.26
10	.6	66.8	49.8	3.18	3.84
10	.7	75.1	52.8	2.38	2.51
10	.8	85.8	46.0	4.47	4.03
10	.9	93.7	41.6	5.42	4.42
10	1.0	98.8	43.4	5.31	4.82

The conclusions drawn from these tests were as follows:

1. The magnesian lime used precipitated the zinc in slightly more than the calculated proportion, especially with 10 per cent Zn solutions.

2. The excess of zinc precipitated beyond the theoretical proportion is doubtless a basic salt, as an excess of zinc oxide is known to form such basic compounds with zinc chloride solutions.

3. The purest precipitate was obtained with half of the theoretical amount of lime necessary, and as the theoretical amount was more nearly approached, the precipitate contained more and more lime and magnesia which had not reacted with the zinc chloride solution to form calcium or magnesium chlorides.

4. The precipitates prepared from 10 per cent Zn solutions were rather difficult to filter, but the precipitates from the 5 per cent Zn solutions were easily filtered. On drying the precipitates and preparing them for analysis, those from the 10 per cent Zn solutions were also sticky and difficult to pulverize.

5. It is possible to precipitate all the zinc from a zinc chloride solution without using an excess of lime.

6. Fine grinding of the lime, or the preparation of a lime paste by slaking, before addition to the zinc solutions, gives more complete reaction; also less washing of the precipitate is necessary to remove calcium chloride. A lime paste was found to be the best and most reliable form in which to add the precipitant.

EXPERIMENT WITH OXIDIZED ZINC ORE.

Although the grade of precipitate obtained in the test with zinc chloride was not satisfactory, on account of the large amount of lime and magnesium compounds left in the precipitate, an attempt was made to precipitate the solution obtained by treating oxidized zinc ore from the Wolftone mine of the Western Mining Co., of Leadville, Colo. The ore was mixed with strong sulphuric acid and calcined at 600° C. in order to break up colloidal silicic acid, iron sulphates, and other compounds. The ore was then leached with water and the resulting zinc sulphate solution was used for a series of precipitation tests. The usual procedure would involve leaching the calcine with calcium chloride solution, followed by purification of the solution with crushed limestone before filtering from the residue, but these steps were conducted separately in order that the precipitates of calcium sulphate, etc., might be examined separately. On that account calcium chloride solution was added to the water solution of zinc sulphate (containing small amounts of other metal sulphates), and the resulting precipitate of calcium sulphate was filtered from the solution. After filtration, ground lime rock was agitated with the solution for about one hour for the purpose of removing iron and aluminum from the zinc chloride solution by hydrolysis. Then this precipitate was filtered and the purified zinc chloride solution was treated with the theoretical proportion of a paste of freshly slaked lime to precipitate zinc hydroxide and regenerate the calcium chloride solution. The zinc hydroxide precipitate was then filtered and dried. In normal operation the reaction of the calcium chloride on zinc sulphate would take place in the presence of the ore, and likewise the purification with lime rock.

The results of this test are contained in Table 30.

TABLE 30.—*Results of treating zinc-sulphate leaching solutions with lime compounds.*

[500 c. c. of solution used in each test.]

$CaSO_4$ precipitate.		$CaCO_3$ purification.			$Zn(OH)_2$ precipitate.								
Weight.	Zn.	Weight.	Zn.	Fe.	Weight.	Zn.	Mn.	Fe.	CaO.	MgO.	Cl.	Time.	Percentage of recovery.
Grams.	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Hr. Min.*	
35......	0.94	14.0	0.94	0.89	21.5	23.8	0.05	0.23	37.4	1.5	6.75	5	44.6
28......	.26	14.0	1.28	1.71	18.0	26.4	Trace.	.47	35.0	2.4	8.90	10	42.3
37......	.94	19.0	.94	1.06	19.5	25.0	0.10	.23	35.2	1.5	4.08	20	43.4
41......	.60	11.5	.77	1.01	22.2	30.8	Nil.	.33	28.4	1.0	5.96	40	60.9
39......	.77	16.5	.69	.91	22.0	34.1	Nil.	.48	26.9	1.3	8.66	1	66.7
41......	.77	15.0	.60	.92	22.5	40.0	Trace.	.47	22.4	0.5	7.81	2	80.2
35......	.77	14.5	.86	1.31	23.5	41.0	2.18	.70	13.5	5.7	6.66	4	85.5
43......	.60	13.5	.94	1.30	20.3	51.7	2.16	.42	6.6	3.5	6.04	8	93.4

From these results the following conclusions can be drawn:

1. The reaction of the lime paste on the zinc chloride solution requires more than one hour before it is approaching commercially satisfactory completion. Four or eight hours' time is better, and the indications are that 24 hours would be still better.

2. The lime content of the zinc hydroxide precipitates is high unless the time of reaction is more than four hours. It is probable that the coating of zinc hydroxide over lumps of calcium hydroxide slows down the reaction of zinc chloride on the lime, making necessary long agitation in order to obtain a complete double decomposition.

3. The preparation of a lime paste of sufficient strength that 1 c. c. is equivalent to 0.2 gram of zinc was found to be easily attainable.

4. The chlorine content of the air-dried precipitate of zinc hydroxide was variable but seemed to tend toward an average of 6 per cent. This chlorine is probably present as basic chloride of zinc. When the hydrate of zinc is ignited to form zinc oxide most of the chloride of zinc would be volatilized and would have to be caught again in water for reprecipitation.

5. The ore that was treated in this series of tests contained manganese, which was precipitated with the zinc, discoloring the zinc

hydroxide to a dirty brown color when the reaction was nearly completed. Manganese sulphate is not hydrolized in the presence of crushed lime rock in cold solutions and hence is not completely removed with the iron and the alumina.

6. The precipitate of calcium sulphate formed in the first reaction of the process carries very little zinc and can be discarded. Not more than 1 per cent of the total zinc is lost in this way.

REMOVAL OF MANGANESE.

USE OF CHLORIDE OF LIME IN MODERATE AMOUNTS.

On account of the discoloration of the zinc hydroxide precipitate with manganous hydroxide and its oxidation products, further tests were necessary in which removal of the manganese by oxidizing it with some form of chlorine was sought. In the first set of tests made, chloride of lime was used to supply the chlorine. Twenty pounds of $CaOCl_2$ per ton of solution were used in each test, and a series of tests were made in which various amounts of sulphuric acid were added to liberate more or less of the chlorine from the chloride of lime. These amounts of sulphuric acid are recorded in the columns of "acid equivalents." The calcium chloride and chloride of lime were added together, forming the first precipitate, which was removed by filtration, then crushed-lime rock was added to remove iron and alumina, followed by precipitation of the zinc with a paste of slaked lime. The results are contained in Table 31.

TABLE 31.—*Results of treating zinc sulphate solutions with lime compounds for removal of manganese.*

[In each test: 20 pounds of $CaOCl_2$ per ton of solution; 500 c. c. of solution; 30.50 grams Zn per liter; time, 4 hours.]

Acid equivalent.	$CaSO_4$ precipitate.		$CaCO_3$ precipitate.				$Zn(OH)_2$ precipitate.						
	Weight.	Zn.	Weight.	Zn.	Fe.	Mn.	Weight.	Zn.	Mn.	Fe.	CaO.	Cl.	Percentage of recovery.
	Grams.	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
0.0	28.6	Trace.	14.7	4.86	3.85	0.16	32.5	38.1	0.07	0.09	21.55	11.82	81.2
.125	46.4	Trace.	22.0	2.50	2.27	.08	30.1	37.1	.04	.22	20.90	9.24	73.3
.25	39.2	Trace.	20.3	2.10	2.53	.14	32.2	36.5	.04	.17	19.75	9.56	77.0
.5	50.3	Trace.	27.5	1.18	1.87	.15	32.5	36.5	.035	.31	20.85	8.51	77.8
1.0	39.0	Trace.	26.3	1.18	1.89	.52	37.8	32.6	.035	.13	20.60	9.66	80.7
1.5	49.9	Trace.	28.3	1.18	1.74	.91	34.1	39.2	.021	.13	19.90	11.78	87.5
2.0	43.0	Trace.	27.0	0.79	1.18	.74	33.9	37.3	.028	.17	19.06	10.25	83.9

The table shows that the manganese content of the final precipitate was low; also it was observed that the precipitate was of a satisfactory white color. The lime content of the precipitate was too high, as insufficient time had been allowed for the reaction of the lime on the zinc chloride solution. On that account further treatment of these precipitates with zinc chloride solutions, to see whether the lime would react and go into solution as calcium chloride, was thought best. This was done with the precipitates from the next set of tests about to be described.

USE OF LARGER AMOUNTS OF CHLORIDE OF LIME.

As the improved color of the precipitate was encouraging, another series of tests was made with greater amounts of chloride of lime, in the hope that most of the manganese could be precipitated as the dioxide, in a marketable product. The results of the tests are reported in Table 32. The use of the greater amounts of chloride of lime seemed to give erratic results, and the precipitates were colored with manganese, especially those in the first and third tests, which were very brown. However, the lower percentage of lime left in the precipitates, as compared with the previous set of tests, was a favorable feature.

TABLE 32.—*Results of treating zinc sulphate solutions with excess $CaOCl_2$ to remove manganese.*

[500 c. c. of solution, with 3.05 per cent Zn content, used in each test; solution agitated 4 hours at 20° C.]

$CaOCl_2$, pounds per ton.	$CaSO_4$-$CaCO_3$ precipitate.				$Zn(OH)_2$ precipitate.							
	Weight.	Zn.	Fe.	Mn.	Weight.	Zn.	Fe.	Mn.	CaO.	MgO.	Cl.	Percentage of recovery.
	Grams.	*P. ct.*	*P. ct.*	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
40	79.2	3.15	2.72	0.75	27.5	45.7	0.91	2.47	13.10	3.85	4.58	82.6
60	101.2	1.05	6.85	.75	26.2	49.7	1.72	.40	9.55	4.47	2.91	85.0
80	93.0	2.63	3.56	.76	27.9	47.2	.18	2.21	15.05	3.96	3.54	86.5

USE OF ZINC CHLORIDE SOLUTION.

The content of lime left in the precipitates was still too large, so the precipitates of this last series of tests were further treated with a 5 per cent $ZnCl_2$ solution, to correspond with the conditions for precipitation of zinc chloride solutions with lime in a series of countercurrent agitators and thickeners. The data on these tests are contained in Table 33. Test 4 was made with a mixture of precipitates, from one of the former sets of tests, having a high lime content.

TABLE 33.—*Results of treating zinc hydroxide precipitate with zinc chloride solution to remove lime.*

Test No.	Original precipitate.						Final precipitate.					
	Weight.	Zn.	CaO.	MgO.	Cl.	Mn.	Weight.	Zn.	CaO.	MgO.	Cl.	Mn.
	Grams.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
1	18.0	45.7	13.10	3.86	4.58	2.47	21.7	54.2	1.29	2.72	8.18	0.13
2	19.1	49.7	9.55	4.47	2.91	.40	26.5	53.1	1.10	2.42	9.00	.39
3	17.0	47.2	15.05	3.96	3.51	2.21	20.8	54.7	1.90	2.50	6.05	.73
4	100.0	34.9	19.52	......	......	......	120.7	49.4	3.40	.40	8.25	.08

This series of tests shows that 4 hours' agitation of the precipitate with fresh zinc chloride solution removes practically all of the lime, leaving less than 2 per cent CaO in the precipitate, although not much of the magnesia was removed. The percentage of zinc was not raised much and the percentage of chlorine was increased. It was noticed that the precipitates tended to combine with the zinc chloride of the solution to form hard cakes, supposedly oxychloride of zinc, which will set somewhat like magnesium oxychloride cement. When the zinc hydroxide is later calcined to form zinc oxide, all of the zinc chloride would be volatilized and could be caught from the calciner gases and returned to the zinc chloride solutions. For some reason the manganese content of the residual zinc hydroxide was lowered markedly, although the precipitate still had a dark color.

TESTS WITH CHLORINE WATER.

As no advantage was to be gained from using large amounts of chloride of lime for preventing manganese from entering the precipitate, it was next determined to try chlorine gas for the oxidation of the manganese, as well as to try the effect of heating the solution while using chloride of lime. The conditions of the five tests made were as follows:

No. 1. Hot, $CaOCl_2$, 20 pounds per ton of solution.

No. 2. 200 c. c. of saturated chlorine water per 500 c. c. of solution, cold.

No. 3. 400 c. c. of saturated chlorine water per 500 c. c. of solution, cold.

No. 4. 200 c. c. of saturated chlorine water per 500 c. c. of solution, boiling.

No. 5. 400 c. c. of saturated chlorine water per 500 c. c. of solution, boiling.

The data on the precipitates obtained in these tests is contained in Table 34. The chlorine was applied to the solution at the same time as the lime rock, as this provides the conditions under which manganese should theoretically be precipitated. Actually very little manganese was eliminated from the solution; later it was thought that the manganese must be present in the oxidized solutions as manganates or permanganates of zinc or calcium, all of which are very soluble. In these tests insufficient lime for complete precipitation was used, as it was hoped that with the smaller amounts of lime zinc hydrate of reasonable purity could be obtained in four hours' reaction. The results indicate that the precipitate can not be kept free of lime unless the lime and the zinc chloride solution are in contact at least eight hours or more. The precipitates obtained in this series of tests were white and contained sufficiently low percentages of iron and manganese to be acceptable. Only the lime and chlorine are left, according to the analyses recorded. Later it was found that sulphates also were present in serious amounts.

TABLE 34.—*Results of treating zinc sulphate solutions with chlorine to oxidize manganese.*

[500 c. c. of 3.05 per cent Zn solution in each test.]

Test No.	$CaSO_4$ precipitate.			$CaCO_3$ precipitate.			Zn $(OH)_2$ precipitate.							
	Weight.	Zn.		Zn.	Mn.	Fe.	Weight.	Zn.	Fe.	Mn.	CaO.	MgO.	Cl.	Percentage of recovery.
		Percentage.	Weight.											
	Grams.		*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*Grams.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
1.....	41.8	0.43	30.5	3.94	Nil.	1.81	12.7	51.7	0.36	0.06	7.6	0.38	6.12	45.0
2.....	42.5	.26	15.2	2.23	2.40	3.30	15.4	46.8	.44	.05	7.3	.25	5.17	43.5
3.....	44.2	.26	19.7	3.67	6.30	2.72	15.2	48.6	.43	.13	8.7	.28	6.81	47.2
4.....	43.5	.87	28.5	3.02	1.20	2.17	13.8	46.0	.45	.24	10.8	.52	6.05	48.6
5.....	45.5	1.22	29.6	11.55	.53	1.85	13.9	49.0	.36	.09	8.3	.38	5.17	41.5

The chlorine used in these tests had been applied as chlorine water, which diluted the zinc chloride solution. In the next series of tests chlorine gas was applied directly to cold solutions.

TESTS WITH CHLORINE GAS.

The designing of this set of tests depended on the following considerations: It had been shown that the treatment with fresh zinc chloride solution for a sufficient length of time would dissolve most of the lime left in the precipitate. Hence a series of cyclic leaches of ore, with purification and precipitation of the liquor during each cycle was designed. The precipitate of each test was partly dewatered and then treated with fresh solution from the succeeding leach, followed by dewatering and treatment with the fresh solution of the next succeeding leach. In this way the zinc chloride solutions were treated by countercurrent agitation and thickening with the lime precipitant. The final precipitate was further washed with water and the wash water was analyzed for zinc. Then the zinc hydroxide precipitate was ignited to form zinc oxide, and to drive off all zinc chloride so that it might be recovered and retreated in solution.

The ore used for these tests was from the Bullion Coalition mine, in the Stockton district of Utah. It was mixed with sulphuric acid in amount equivalent to the zinc in the ore, and was calcined at 700° C. in order to break up all colloidal silica, iron sulphate, and aluminum sulphate, leaving only zinc sulphate to be leached with the calcium chloride solution returned from precipitation of the previous cycle. In this way calcium sulphate was precipitated in the pulp. Then chlorine gas was bubbled through the pulp till it smelled strongly, after which ground calcium carbonate was added and the pulp was agitated for two hours to cause hydrolysis and precipitation of iron, aluminum, and manganese. The pulp was then ready for filtration and precipitation.

The data on these cyclic leaching and precipitation tests is contained in Table 35. The high percentage of manganese in the second precipitate was thought to be due to the use of lime in excess of the theoretical proportion. In nearly every test the solution was agitated for about four hours after the lime was added, but in the eighth and ninth tests only about two hours' time was allowed, which accounts for the higher proportion of lime in the precipitates from these two tests. The chlorine content in the precipitates varied considerably as they tended to crack and shrink while on the filter, with the result that good washing was difficult; hence various amounts of zinc chloride were left in them. It is possible that repulping of this precipitate with water, followed by a second filtering would be necessary.

The precipitates were also analyzed for sulphates, and surprisingly large amounts were found. Later, it was determined that the amount of sulphates present in the final precipitate was very nearly equivalent to the amount of gypsum that can be dissolved in the zinc chloride solution. The remedy would be to use solutions containing higher proportions of zinc and also to use a slight excess of calcium chloride in order to lower the solubility of the calcium sulphate due to the "common ion" effect.

After drying, weighing, and analysis, the precipitates were ignited at a low red heat, in order to form zinc oxide and to drive off any zinc chloride present. After burning, most of the precipitates had a yellow tinge. The analysis of the ignited oxide shows that it was still considerably contaminated with calcium sulphate.

TABLE 35.—*Results of cyclic leaching and precipitation tests.*

[Test performed May 1, 1916, by C. W. Frith and O. E. Young.]

Precipitate No.	Quantity of solution used.	Quantity of zinc in solution.	Weight of precipitate.	Analysis of precipitate.								Zinc content of precipitate after burning.	Zinc volatilized.	Barren solution.			Wash water.			Total zinc recovered.
				Zn.		Fe.	Mn.	CaO.	MgO.	Cl.	SO₄.			Quantity.	Zinc content.		Quantity used.	Zinc content.		
				Per cent.	Grams.										Per cent.	Grams.		Per cent.	Grams.	
	C. c.	*Gms.*	*Gms.*			*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*C. c.*			*C. c.*			*Gms.*
1	500	16.6	31.0	50.8	15.8	0.75	0.207	2.31	None.	3.61	7.7	67.1	7.3	450	0.25	1.12	100	1.62	1.62	18.54
2	500	16.6	42.0	50.4	21.2	.12	1.54	3.96	None.	17.05	1.65	65.1	11.05	400	.05	.200	138	.82	1.14	22.54
3	500	16.6	10.0	53.0	5.3	.18	.27	3.26	None.	9.76	5.70	67.5	7.9	480	.15	.720	102	.87	.89	6.86
4	500	16.6	21.8	49.8	10.8	.15	.208	4.6	None.	7.45	12.50	62.8	5.8	360	.115	.417	250	.785	1.96	13.18
5	500	16.6	27.0	51.0	13.8	.28	.54	4.08	None.	12.95	4.69	65.2	9.45	430	.10	.43	124	.95	1.18	15.41
6	500	16.6	29.65	55.7	16.5	.20	.234	2.32	None.	11.00	3.08	71.4	6.90	600	.05	.30	173	1.00	1.73	18.53
7	500	16.6	20.30	53.60	10.9	.23	.207	3.20	None.	9.26	7.95	67.6	7.05	575	.30	1.72	157	.75	1.18	13.81
8	500	16.6	27.6	48.3	13.3	.89	.055	7.17	None.	8.42	10.40	58.4	3.00	207	.125	.32	210	.58	1.22	14.84
9	500	16.6	22.70	45.0	10.2	.092	.134	6.76	None.	7.05	13.70	63.7	2.00	550	.30	1.65	210	.62	1.30	13.15
10	500	21.6	54.6	52.0	28.4	.04	.268	4.87	None.	10.60	12.30	64.5	7.50	1,090	.25	2.72	200	1.00	2.00	33.12
11	500	21.6	....	....	....	....	....	....	....	....	....	....	....	680	1.50	10.20	....	....	....	10.20
12	500	21.6	....	....	....	....	....	....	....	....	....	....	....	1,215	2.40	29.16	....	....	....	29.16

Total quantity of Zn in the solution was 214.2 grams; total quantity of zinc in the precipitate and in the solution after the precipitate was filtered off was 209.34 grams.

From this set of tests the following conclusions may be drawn:

1. It is possible to treat ores with calcium chloride and sulphuric acid to obtain zinc chloride solutions containing only small amounts of dissolved calcium sulphate.
2. These solutions can be purified of iron, aluminum, and manganese by the use of chlorine gas followed by the use of crushed lime rock, which causes hydrolysis of the highly oxidized compounds of these metals, the generation of calcium chloride during hydrolysis precipitating some of the calcium sulphate dissolved in the zinc chloride solution.
3. Precipitation with lime gives a hydrate of zinc and regenerates calcium chloride for use in the next cycle.
4. The main impurities of the zinc hydrate are excess calcium hydroxide, calcium sulphate, calcium chloride and zinc chloride. More than 8 hours' contact of solution with the lime is necessary for converting all the calcium hydroxide to the soluble calcium chloride. Careful washing will remove most of the calcium chloride and much of the zinc chloride. During ignition to the form of oxide, most of the remaining zinc chloride is volatilized. The content of calcium sulphate can be kept down by using solutions containing more than 5 per cent zinc and by adding an excess of calcium chloride to the zinc chloride solution.
5. About 6 per cent of the total zinc content in the zinc hydroxide precipitate is present as zinc chloride, which is volatilized during ignition to the oxide.
6. With 3.5 per cent zinc solutions the final zinc oxide product will average over 65 per cent zinc.
7. No filtration or thickening difficulties are encountered.

BISULPHITE SOLUTIONS.

The employment of bisulphite solutions as a means of recovering zinc has attracted some attention in times past, so that the literature is full of speculations and arguments as to the advantages of sulphurous acid leaching of zinc, but to date no one has been able to handle the precipitation problem satisfactorily.

It has been proposed to recover the zinc as the monosulphite from such solutions by heating them to a sufficient temperature to drive off the extra sulphur dioxide. This sulphur dioxide was to be used over again in the leaching step of the process. Tests of sulphurous acid leaching made at the Salt Lake City station and of sulphurous acid as a solvent are described in another part of this bulletin. Practically all of the zinc in the bisulphite solutions could be precipitated as insoluble zinc monosulphite by boiling, but the solution had to be boiled till more than half of the water had been evaporated. Further, the zinc bisulphite solutions oxidized rapidly, so that the solutions tested contained nearly 50 per cent of the zinc as sulphate.

As the small-scale laboratory tests probably afforded a better chance for air to act on the surface of the solution than would probably be found in practice, the tests were repeated, care being taken to prepare a fresh solution which had had a minimum contact with the air. Even then 33 per cent of the zinc oxidized to the sulphate condition before the precipitation tests by boiling could be performed. On that account, in any process involving the precipitation of zinc bisulphite solutions provision must also be made for precipitating zinc sulphate. This necessity has led some to propose evaporating the solution to dryness and recovering only zinc sulphate, which could be sold as such or calcined to form sulphur dioxide and zinc oxide. The cost of evaporation would be a serious objection to such a method, as not more than a 10 per cent zinc solution can be prepared and the amount of water to be evaporated is too large.

Also, tests were made of the method of precipitating such solutions by the addition of enough zinc oxide from a later stage of the process to neutralize the acidity due to sulphurous acid and to the bisulphite of zinc. This would form zinc monosulphite, which is insoluble and should precipitate. On filtering the precipitate from the solution and calcining it, the zinc monosulphite breaks up into sulphur dioxide and zinc oxide and no solution has to be evaporated. This method worked very well with the zinc bisulphite solutions, but zinc sulphate solutions would not precipitate. Consequently, this method would not meet the requirements mentioned of having a precipitant which would recover the zinc from both zinc bisulphite and zinc sulphate solutions, but was preferable to heating the solution of zinc bisulphite.

Ammonia will precipitate zinc from either sulphate or bisulphite solutions, after which the ammonia can be recovered from the precipitated solutions by boiling with lime. However, absorption of ammonium compounds by the precipitated zinc hydroxide, together with mechanical losses of the ammonia, might make this process unattractive. Sodium hydroxide will also precipitate the zinc from both forms, but is too expensive.

SUMMARY.

Summarizing, the precipitation of zinc from its various solutions constitutes the major problem in the hydrometallurgy of zinc. Electrolytic precipitation is only being perfected and the installation costs of an electrolytic zinc plant are so great that such a process could be used only for the treatment of large bodies of ore or in a custom plant. Chemical precipitation of a zinc oxide product from most of the solutions that are capable of commercial preparation is difficult. Zinc chloride solutions are easily precipitated by lime, although the product is not of the highest grade. Sulphate solutions of zinc can not be precipitated economically by any known direct precipitation process, but can be precipitated after conversion to zinc chloride. The sulphite solutions of zinc deserve further study.

OXIDIZED ZINC ORES.

INTRODUCTION.

Beneficiation of low-grade oxidized ores of zinc has long been a baffling metallurgical problem, and satisfactory methods of treatment have still to be worked out. Most ores of this type are ocherous and earthy and can not be concentrated mechanically. On that account such ores are not worked for their zinc content alone unless they are of sufficiently high grade for direct shipment to the smelter. Throughout the intermountain western region are deposits of oxidized zinc ore containing as high as 25 per cent zinc which can not be concentrated by any known method and is rarely accepted by zinc smelters. For a time, shortly after the European war sent the price of zinc up, ores with as low as 20 per cent zinc content were shipped from the intermountain States to the zinc smelters in the gas belt, because the smelters did not have sufficient roaster capacity and wished oxidized ores in order to keep the distillation furnaces going at full capacity while the roasting equipment was being augmented.

CONCENTRATION PROCESSES.

GRAVITY CONCENTRATION.

As mentioned, the ores containing oxidized zinc minerals are usually too ocherous and clay-like to permit mechanical separation. A few ores are relatively "free" and can be concentrated by jigs, tables, and other gravity concentration machines. The calamine ores of Arkansas, which are being concentrated and shipped to zinc smelters, are an example of this type of "free" ore. Ores containing 2 to 10 per cent zinc are the ones usually mined and milled in this way, the Joplin type of wet mill being in favor. Needless to say, all of the zinc mineral in the more finely divided ore is lost, and the slime problem is important; flotation has not been satisfactory. Hence, recoveries by gravity concentration vary from 40 to 70 per cent, probably averaging about 60 per cent. The oxidized zinc ores of the intermountain western region are usually too intimately associated with the gangue material to be amenable to this type of treatment. In Arkansas and Missouri the oxidized zinc ores were formed from the oxidation, in place, of blende coarsely disseminated in a flint or chert gangue, whereas those of the intermountain region are from oxidation of contact metamorphic deposits or of replacement deposits and are usually contaminated with the oxidized minerals of iron, lime, magnesia, manganese, and other metals. Much of this ore is so yellow with limonite that individual particles of smithsonite or of calamine are very difficult to distinguish. It is these ores that resist gravity concentration.

FLOTATION.[a]

As in most western oxidized zinc ores the minerals are finely intercrystallized, concentration of such ores by fine grinding and flotation seemed possible. Sulphidizing and flotation has become popular for the treatment of oxidized ores of lead and of copper, and it was thought the same method might be applied to oxidized zinc ores.

However, in none of the ores tested could any apparent concentration be made of the zinc carbonate and zinc silicate minerals by the usual sulphidizing methods. In fact the flotation froth from the yellowish ocherous ores from Utah, Nevada, and Colorado was often slightly lower in zinc than the tailing. Sulphureted oils, ammonium sulphide, and other such reagents all failed.

Mr. Frank Bird, of Salt Lake City, reported that by sulphidizing with sodium sulphide followed by the use of oleic acid as a flotation oil, he was able to obtain a partial flotation of oxodized zinc minerals. By making up a synthetic ore of zinc carbonate and quartz it was possible to verify his tests, but when the method was applied to a number of natural ores, high in lime, the result was that the lime was floated and the zinc minerals were left behind with the other gangue minerals. Supposedly a superficial film of metallic oleates is formed in the presence of sodium hydroxide, and these oleates are easily "oiled" by the flotation oil. Possibly resinates could be used for the same purpose, as is probably being done at the plant of the Chino Copper Co. at Hurley, N. Mex., where copper carbonates are treated with a solution containing sodium sulphide, sodium resinate, etc. However, commercially valuable results in the flotation of zinc carbonate by any of these methods were not obtained in the tests at the Salt Lake City station.

Where a small amount of lead carbonate was present with the zinc carbonate, usually the lead carbonate could be sulphidized and floated in the presence of the zinc, providing no manganese dioxide, lead peroxide, basic sulphates of iron, or other elements deleterious to sulphidizing were present.

MAGNETIC SEPARATION OF LIMONITE AND OXIDIZED ZINC MINERALS.[b]

Magnetic separation of limonite in oxidized ores of zinc was deemed possible because a very slight reducing roast is reputed to render limonite "magnetic," so that it could be removed from the ore with an ordinary low intensity magnetic separator.

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[a] Experimenters: R. W. Johnson and A. J. McChrystal.

[b] Experimenters: S. S. Arentz and O. C. Ralston.

Tests were made of a sample of ore from the Bullion Coalition mine. The results of an ultimate analysis of this ore are given in the following table, the mineralogic composition, as calculated from this analysis, being included.

Composition of ore from Bullion Coalition mine.

Ultimate analysis.		Mineralogic composition (calculated).	
	Per cent.		*Per cent.*
Insoluble	12.5	Kaolin	22.02
CaO	1.1	Calcite	1.97
Fe	15.0	Smithsonite	44.65
Al_2O_3	2.6	Calamine	4.65
Zn	30.4	Anglesite	.31
Pb	.21	Cerrusite	.00
Cu	.13	Pyrite	2.25
S	.92	Limonite	22.65
CO_2	19.04	Malachite	.23
Ag	[a].15		
		Total	99.75

[a] Ounces per ton.

In appearance the yellow ocherous limonite predominated and seemed to be plastered over the particles of zinc carbonate. The latter were occasionally visible as large white specks, but in the greater part of the ore were indistinguishable.

The material was crushed to pass a 20-mesh screen, mixed with 5 per cent of coal dust, and heated in a muffle furnace out of contact with the air for two hours. The weight of the resulting calcine was only 85.5 per cent of the original weight. Of this calcine 40 per cent was separated on a low-intensity Campbell magnetic separator. The magnetic material analyzed 22.3 per cent zinc. The other 45.5 per cent of the ore contained 38.2 per cent zinc, a product which could easily be sold, as prices for oxidized zinc ore are figured on a basis of 35 per cent zinc. Normally a 30 per cent zinc ore can not be shipped from Utah to the "gas belt" at a profit on account of freight costs.

With prewar conditions and 5-cent spelter, the net value in Salt Lake City of 1 ton of "carbonate" ore containing 30 per cent zinc would vary from nothing to $5, whereas that of a ton of 38 per cent zinc ore under the same conditions would be about $12.50. As mining costs have to be paid out of this net return and the higher grade ore has a sure market, while the price paid for the lower grade ore fluctuates according to the conditions of the stock piles at the zinc smelters, such ore could easily be treated at a profit by magnetic separation, although the reject contains 22.3 per cent zinc—30 per cent of the total zinc in the ore. The low recovery of zinc, however, would normally condemn such a procedure.

Many small-scale tests were made in order to determine the effects of various percentages of reducing agent, different lengths of time, and different temperatures of reduction. The averages of the best results showed that on nearly all different sizes it was possible to obtain a nonmagnetic residue containing 40 per cent zinc and weighing about 42 per cent of the original ore, and a magnetic product containing 21.8 per cent zinc and also weighing about 42 per cent of the original ore. This latter work was done with a sample containing only 24.82 per cent zinc.

None of the zinc in the magnetic product could be separated, on account of the intimate crystallization of the zinc with the iron minerals. The ore used is representative of much of the oxidized brown "carbonate" ore of the Rocky Mountain mining districts.

CALCINING OXIDIZED ZINC ORES.[a]

Calcining oxidized zinc ores as a method of concentration has been repeatedly proposed. The fundamental principle of this treatment is that the water of hydration and the combined carbon dioxide in the ore can be driven off, leaving all the zinc and the other metals in a product of much less weight and consequently higher grade. There is no doubt that calcining any zinc ore which is normally of shipping grade, but which is at some distance from the zinc smelters, will pay. For instance, with the oxidized zinc ores of Nevada, the freight rates to the "gas belt" are $8 to $11 per ton; hence a loss in weight by the ore amounting to 25 or 30 per cent effects an important saving.

A number of such ores were calcined in the laboratory by roasting 100-gram samples in dishes of 5-inch diameter at 800° C. for one to two hours. The roasted samples, on standing in the laboratory for a number of days, did not take up moisture and carbon dioxide to any serious extent, as shown by the figures in Table 36, but in some samples the lime slaked vigorously when they were wetted with water. Therefore such calcined ore would have to be shipped in closed cars to keep it dry and prevent its increasing in weight again. As the normal condition of transportation is such that the railroads prefer loading eastbound zinc ore in box cars, this would be no disadvantage.

The figures in the table show that few nonshipping ores would be raised to shipping grade, but that with the shipping ores important savings in freight would be effected and a slight concentration of the zinc value, which on some contracts would more than pay for the cost of calcining.

The analyses of these ores are found in Table 3 (p. 21).

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[a] Experimenters: H. J. Morgan, H. C. Neeld, and W. R. Sholes.

TABLE 36.—*Results of calcining oxidized zinc ores.*

[100-gram lots calcined 2 hours at 800° C.]

Mine and sample.	Before heating.		After heating.		Weight at—			
	Zinc.	Lead.	Zinc.	Lead.	Beginning.	7 days.	12 days.	16 days.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Grams.*	*Grams.*	*Grams.*	*Grams.*
Scranton, No. 1	25.6	11.7	31.4	11.0	82.4	83.0	83.7	83.8
Scranton, No. 2	8.0	....	10.9	....	83.4	84.5	85.3	85.6
Scranton, No. 3	17.9	....	22.0	....	77.8	80.3	81.3	81.8
Lower Mammoth, No. 1	9.0	....	8.9	....	84.9	84.9	84.9	84.9
Lower Mammoth, No. 2	11.4	....	11.4	....	86.1	86.1	86.1	86.1
Lower Mammoth, No. 3	12.0	....	12.0	....	85.5	85.5	85.5	85.5
Bullion Coalition, No. 13	25.5	....	30.5	....	82.2	82.2	82.3	82.4
Yellow Pine, No. 1	31.6	17.6	38.9	18.0	76.5	76.8	77.1	77.3
May Day, No. 4	23.1	13.9	26.7	12.6	82.3	83.9	85.1	85.2
Lake View, No. 104	23.3	26.2	27.8	29.2	79.8	79.8	79.9	79.9
Lake View, No. 106	28.2	....	38.0	....	77.9	80.6	83.1	84.3
Lake View, No. 107	9.3	13.6	16.6	17.0	67.4	67.7	67.9	67.9
Santaquin King	44.7	....	47.7	....	92.0	92.0	92.0	92.2
Western Mining Co	20.0	....	22.4	....	82.0	82.0	82.1	82.1

IGNEOUS CONCENTRATION.[a]

By igneous concentration of oxidized zinc ores is meant that process in which the ore is mixed with an excess of fuel and heated under forced draft so that zinc distills out of the ore and reoxidizes in the air to form zinc oxide fume, which is later filtered from the flue gases. Most of the commercial zinc oxide in the United States is made in this manner, although the mechanical equipment used for the process varies widely. The increased use of zinc oxide in paints, for filling rubber, and other purposes, make this process of importance. It is said that some automobile tires contain 60 per cent ZnO. Further, the process seems to offer a solution of the problem of treating low-grade oxidized ores of zinc which are amenable to no other concentration process.

WETHERILL GRATE FURNACE.

The best known furnace in which zinc oxide is made from oxidized ores is the Wetherill[b] grate furnace, which is built much like the fire box under a large stationary boiler. The mixed ore and fuel (coke breeze or anthracite coal slack) is dropped on the grates in a bed 4 inches to 12 inches deep. The air is blown in under the grates and thus keeps them relatively cool. The ore bed is heated to 1,100° to 1,300° C. by the combustion of part of the fuel; at this temperature the zinc is reduced and distilled from the ore particles but reoxidizes in the excess of air which is blown into the furnace. After partial cooling of the gases they are filtered in a bag house to remove the fume of zinc oxide. The grates differ somewhat in design, some being hollow and filled with holes from which the air is blown into the mass, and others being like ordinary boiler grates. In nearly all of them the air openings are tapered, with the largest diameter at the bottom in order to prevent blinding. The grates are charged and discharged by hand, so that the cost of labor involved is rather large. In addition the grates tend to buckle and have to be removed periodically from the hot furnaces and replaced by fresh grates. The buckled members are heated uniformly in a special furnace and then straightened.

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[a] Experimenters: H. C. Neeld, J. F. Cullen, R. H. Bradford, and O. C. Ralston.

[b] Stone, G. C., Oxide of zinc: Bull. 129, Am. Inst. Min. Eng., September, 1917, pp. 1217-1228.

Another type of furnace used is very similar to the ordinary zinc retort furnace except that an air pipe is led to the back of each muffle and zinc oxide is driven off instead of metallic zinc. This furnace likewise requires much hand labor, and, being a retort furnace, has a rather high fuel consumption.

A rotary cement kiln is reported to have been used as a furnace at a plant in Mexico. The ore was fed continuously into the cool end and discharged at the hot end, and a reducing atmosphere was maintained. This distilled all the zinc, which reoxidized in the flue gases. The fuel consumption was said to amount to about 15 per cent of fuel oil based on the weight of the charge. This method has the advantage over the two previously described methods that all the fuel is used in the flame that heats the furnace, so that a reducing atmosphere is easily obtained. The writers have not been informed whether the zinc oxide product was sufficiently pure for use as pigment. However, evidently air could be mixed with the hot gases in a combustion chamber at the head of the kiln in order that all carbon might be burned and the zinc oxidized completely. Rotary kilns often make much dust, but most of this could be caught in a settling chamber, so that the zinc oxide product need contain no more dust than that made on Wetherill grates. For making pigment the kiln would have to be fired with either oil or gas and not with powdered coal, because the ash would contaminate the zinc oxide. Ash contamination might not be serious if the purpose was to make a high-grade concentrate for the zinc smelter rather than pigment. The only disadvantage of this type of furnace for igneous concentration of zinc ores is in the power necessary to turn a heavy kiln.

A blast furnace would be a much better type for continuous mechanical feeding and firing, and would use most of the power for the blast. A number of attempts have been made[a] to blow zinc oxide in blast furnaces, but one great drawback to the method has been that a slag has to be formed to facilitate removal of the residue. When the usual types of slags were made the zinc oxide tended to enter the slag instead of entering the fume. One of the latest attempts was that of R. D. Divine and B. F. Hedges to recover zinc in a lead blast furnace at the old South Chicago plant of the American Smelting & Refining Co.[a] The material treated was a zinciferous lead-smelting slag. Only 60 per cent of the zinc could be blown before the slag melted and ran out of the furnace. A similar plant,[b] making similar recoveries, is now in operation at Florence, Colo.

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[a] Ingalls, W. R., Zinc burning as a metallurgical process: Bull. 129, Am. Inst. Min. Eng., September, 1917, pp. 1229-1234.

With these results in mind, some experiments in the blast furnace treatment of such ores were made at the Salt Lake City station, with the object of blowing more of the zinc into the fume, if possible. A small "homemade" blast furnace was constructed from fire brick and sheet iron, as shown in figure 10. The three tuyères were made of iron pipe, and as they burned off they were pushed farther into the furnace. The pipes leading to the tuyères from the air main were made of 2-inch canvas hose, so that they could be pinched down till the right amount of air was being used. The fire-brick lining of the furnace melted away several times during the course of about 25 runs, but was easily replaced and served the purpose, although a standard water-jacketed lead blast furnace would be needed for large-scale work. The inside dimensions of this furnace were about 5½ feet in height, 1 foot in diameter at the top, and 18 inches in diameter at the bottom to prevent hanging. After heating the furnace with any convenient type of fuel the charges were added with various percentages of coke fuel. In each test the residue was drained out of the furnace before another charge of different composition was used. The slag analysis served as an index of the completeness of volatilization of the zinc from the ore. Under these conditions such a furnace could easily melt 2 or 3 tons of ore daily, although it was not run continuously for more than a few hours, a

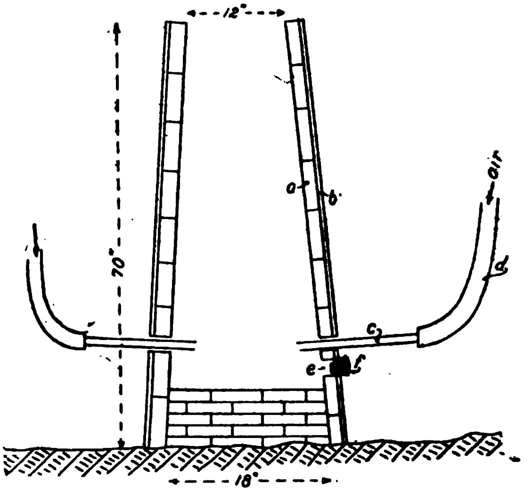

FIGURE 10.—Laboratory blast furnace used for zinc-oxide experiments. *a*, Fire-brick lining; *b*, sheet-iron casing; *c*, 1¼-inch iron pipe; *d*, 2-inch canvas hose; *e*, slag notch; *f*, fire-clay plug.

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[a] Pulsifer, H. G., Zinc oxide from lead blast-furnace slag, South Chicago: Met. and Chem. Eng., vol. 13, Nov. 1, 1915, pp. 783-785.

[b] Hall, R. G., Some economic factors in the production of electrolytic zinc: Bull. 129, Am. Inst. Min. Eng., September, 1917, pp. 1287-1302.

wheel-barrow load being used to a charge, and only in the daytime. On that account the fuel consumption is doubtless too high, varying from 25 to 33 per cent of coke, based on the weight of the ore.

At first the standard lead-smelting slags containing various proportions of iron, silica, and lime were aimed at, but the recoveries of zinc were low, varying from 10 to 50 per cent. Slags so calculated that an excess of one of the slag-forming constituents over the others was used gave much better results. If lime or silica were used in excess, 80 to 95 per cent of the total zinc was obtained, but the slag had a much higher melting point, and more fuel was needed to make it pass from the furnace. Presumably the use of slag with a high melting point permits all the zinc to distill from the ore before the gangue is fluxed; thus, the slag has no opportunity to take up zinc oxide. The use of a high percentage of lime might well be expected chemically to displace zinc from the slag, but the successful use of a high percentage of silica can be explained only on the assumption that the formation temperature of the slag is raised above the boiling point of zinc. In fact, one slag containing 67 per cent SiO_2 was formed during a series of tests with increasing amounts of silica. This slag was so sticky that the tap hole had to be drilled and the slag removed by "balling" on the end of a cold iron rod thrust through the hole.

The results of some of these tests are tabulated in Table 37. Recoveries were low with the high-iron slags, but good recoveries were obtained with the high-lime, high-silica, or high-alumina slags. Slags which are unusual in lead, copper, or iron blast-furnace work were formed in this small furnace and removed with very little trouble. The small size of the furnace permitted quick adjustment of the air blast when the slag seemed in danger of sticking; also sticky slag could be removed by balling whenever the furnace was freezing. However, the slag need not be unbalanced to the danger point in order to get as much as 85 per cent recovery of the zinc.

TABLE 37.—*Results of igneous concentration of oxidized zinc ore in a small blast furnace.*

Slag analysis.					Zinc recovery.	Remarks.
SiO_2.	CaO.	FeO.	Al_2O_3.	Zn.		
Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	
34.0	15.0	30.0	....	10.0	50	Charge finely crushed; slag fluid.
46.2	14.4	17.9	12.0	6.4	67	Slag ran freely.
42.0	19.0	21.0	8.0	4.6	75	
39.0	24.0	15.0	8.4	6.4	73	Slag very fluid.
39.0	26.0	13.0	7.0	5.0	76	
35.4	34.7	15.0	6.9	4.7	83	
40.0	26.9	13.7	11.6	6.0	74	
50.0	29.8	9.0	8.1	2.5	88	
51.4	17.8	13.4	12.2	2.5	88	
48.0	19.0	11.0	15.9	2.4	85	
52.0	13.0	12.0	18.0	1.7	84	
67.0	6.2	11.0	16.9	3.5	60	Furnace froze.

Some of the most difficult zinc ores to handle by present commercial methods are the partly oxidized ores in which only the sulphide of zinc can be concentrated. The results of four tests of oxidized ores containing some zinc sulphide are recorded in Table 38. These indicate that the presence of as much as 10 per cent sulphur does not affect the zinc recovery. The ore used was from the Daly-Judge mine at Park City, Utah, and in addition to zinc sulphide also contained sulphide of lead. No lead collected in the bottom of the furnace, and the final slag contained only traces of that metal; consequently, the lead went into the zinc oxide and the product made would be a "leaded" pigment. A serious drawback to making pigment from such ore by this method is the formation of sulphur compounds, which contaminate the product. On the other hand, in making a zinc concentrate, the presence of lead in the product would be a disadvantage. Hence, for making pigment, a preliminary roasting to drive off the sulphur before igneous concentration might be necessary, whereas if the product were sold as zinc concentrate, the smelter would not pay full value for the zinc on account of the lead present.

Some explanation of the figures in Table 38 is necessary. In all these tests the zinc assays were obtained by decomposing the slag samples with a mixture of nitric and hydrochloric acids, followed by the use of hydrofluoric acid to break up ferrates of zinc. However, only hydrochloric and nitric acids were used in determining the "insoluble", which is not the "absolute silica;" hence, the slag analyses are largely of comparative value only.

TABLE 38.—*Results of igneous concentration of low-grade sulphide ores of lead and zinc in a small blast furnace.*

Slag analysis.					Sulphur content of charge.	Zinc recovery.	Remarks.
Insoluble.	CaO.	FeO.	Al_2O_3.	Zn.			
Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	
85.0	3.6	6.2	3.4	1.0	11.4	88	Furnace froze.
87.0	3.0	3.7	8.0	2.0	8.5	81	Slag removed with bar.
67.6	11.0	8.7	8.3	1.7	8.5	87	Do.
91.0	2.0	7.8	....	1.0	8.5	90	Do.

The experience of the River Smelting Co., at Florence, Colo., as mentioned previously, has been that in smelting zinc ore in a blast furnace accretions of zinc oxide and "blue powder" tend to form on the walls of the furnace above the smelting zone. Such masses are so hard to bar down while the furnace is running that the practice is to shut down the furnace every ten days or so for removing the accretions. This method is, consequently, rather more expensive than is desirable. The tests at the Salt Lake City station were not sufficiently extended to observe this trouble.

In order to obtain an idea of the composition of the zinc oxide product and its physical characteristics, several methods were used in collecting it. A wire screen placed across the throat of the blast furnace collected material that contained only about 67 per cent zinc and was contaminated with flue dust. A stream of the furnace gases by-passed through a cold pipe dropped part of its load as a 73 per cent zinc product. Gases from the discharge of the blast furnace sucked through an electrical precipitator comprising a single 4-inch iron pipe to which was applied 18,000 volts, could be almost entirely cleared at velocities ranging up to 18 feet per second, although most of the time a rate of not more than 10 feet per second was maintained. The product, a fluffy white material containing 77.6 per cent zinc, had a slight bluish tint, possibly due to the presence of small amounts of carbon or of some unoxidized zinc (blue powder). There seems to be no reason why this product could not be used for pigment.

The value of the igneous concentration method for concentrating low-grade oxidized ores of zinc may be estimated from the following two representative contracts, which are commonly possible in the marketing of oxidized zinc ores in Utah. With these two contracts as a basis the possible profits from a plant using the process have been calculated.

Oxidized zinc ores are sold at prices calculated on base prices for 35 per cent zinc ore delivered at the smelter. The base prices for 35 per cent zinc ore are as follows: $17 a ton when spelter is 5 cents per pound, $29 when spelter is 8 cents, and $50 when spelter is 15 cents. The base price changes $3 per ton for every change of 1 cent per pound in the price of spelter between 8 cents and 15 cents, and $4 for every change of 1 cent in price between 5 cents and 8 cents. For ore containing less than 35 per cent zinc, a penalty of $2.50 per unit is charged for each 1 per cent deficiency in zinc as far as 30 per cent, and $2 per unit for each deficiency of 1 per cent below 30 per cent zinc content. A credit of $1 per unit is allowed for each 1 per cent of zinc the ore contains in excess of 35 per cent. The freight rate to the "gas belt" where the zinc ores are smelted is $7.50 per ton of ore.

From these figures the net values of oxidized zinc ores in Salt Lake City have been calculated and plotted as shown in figure 11. It can be seen that with normal 5-cent spelter the lowest grade of ore which can be shipped from Salt Lake City to the "gas belt" is about 31.5 per cent zinc ore. With spelter at 8 cents the lowest grade possible is 26.2 per cent zinc ore. Contracts vary somewhat, so that the minimum zinc content for different contracts may vary 2 or 3 per cent zinc in either direction from the figures given.

For the sale of zinc oxide concentrate the scale of ore values given might be used in calculating the value of the product. The net value in Salt Lake City of such concentrate for different percentages of zinc and for different market prices of spelter has been calculated and plotted on the solid-line curves in figure 12. As the credit for each extra unit of zinc above 35 per cent zinc is only $1, even when spelter is selling at 10 cents, the contract price for the zinc oxide when spelter is selling at 7 cents or better is too low.

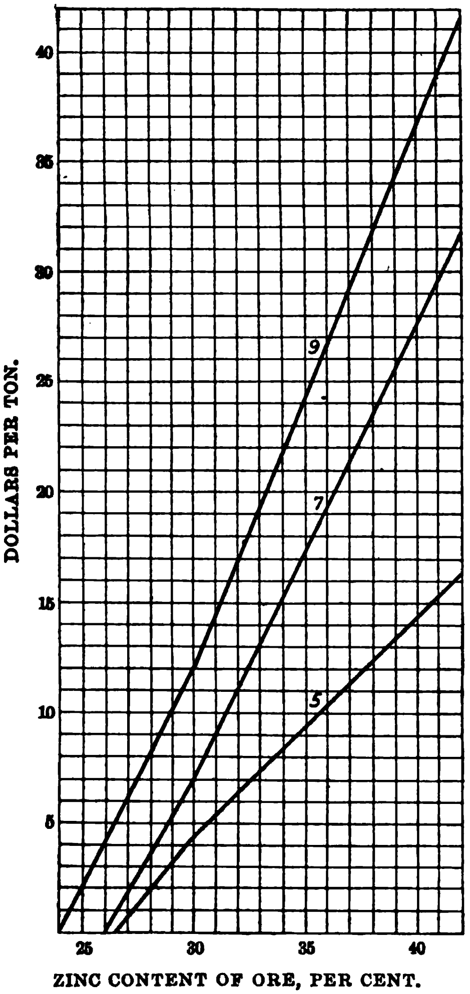

FIGURE 11.—Curves showing net smelter prices for oxidized zinc ores at Salt Lake City, for ores of varying zinc content. Figure near each curve shows price in cents per pound for spelter based on "35 per cent zinc" ore.

A better offer is one made tentatively by a zinc-smelting company which said it was willing to pay for a ton of zinc oxide product containing 75 per cent zinc a base price of 48.75 per cent of the price of a ton of spelter on the day of sale. Thus an 80 per cent zinc product would bring 80/75 of the base price and a 70 per cent zinc product, 70/75 of the base price. The value under these terms of zinc oxide concentrate for varying conditions is also plotted in figure 12 for comparison with the rates paid under the ore contract mentioned. A comparison of the sale prices of zinc oxide under these two contracts shows that with 5-cent spelter the ore contract offers the best price, but with spelter at 7 cents or more the oxide contract offers the best price. In fact, the difference between the two contracts is so marked that careful arrangements would have to be made in regard to the sale of the zinc oxide before commercial production by the process proposed could be undertaken. The zinc smelting company making the offer mentioned wished to have a trial carload of the oxide product before it would close the contract, and thus might bid lower for the next car.

If the oxide product were acceptable as pigment, as it should be, it would probably bring a higher price than is shown by either of the two methods used, but this consideration was eliminated in making the calculations, in order to show only the minimum possible returns.

The cost of a plant for making zinc oxide product would, of course, be dependent on the daily tonnage of ore proposed to be treated. On the supposition that anyone interested in this type of plant would first be concerned in building a small unit in order that every doubtful point might be tested out, the necessary data for estimating the cost of a plant capable of treating 25 tons of ore per 24 hours is presented. The two most important figures that need to be known in order to design such a plant are the volume of air used and the rate of cooling of the hot gases. The figures chosen represent the average amount of air commonly used in lead blast furnaces when they are run with hot tops, and the average rate of cooling of gases in iron flues at a number of smelters. Thus the assumption is made that 200 cubic feet of air per minute is needed per square foot of furnace area measured at the tuyères, and that each square foot of

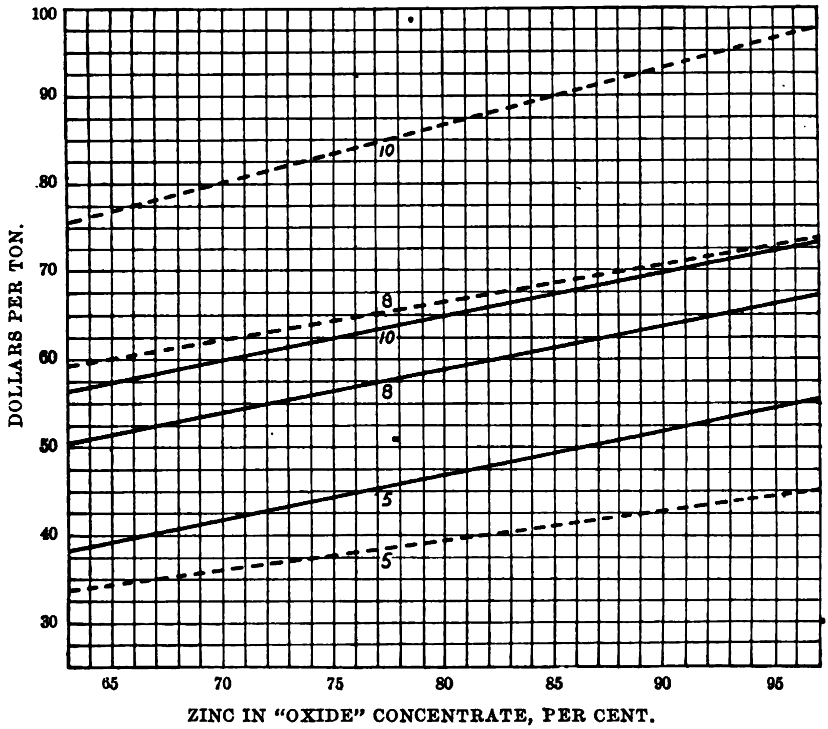

FIGURE 12.—Curves showing change in value of zinc oxide concentrate with varying zinc content. Solid-line curves are based on "ore" contract; dotted-line curves on "zinc oxide" contract. Figure near each curve shows price in cents per pound for spelter based on "35 per cent zinc" ore.

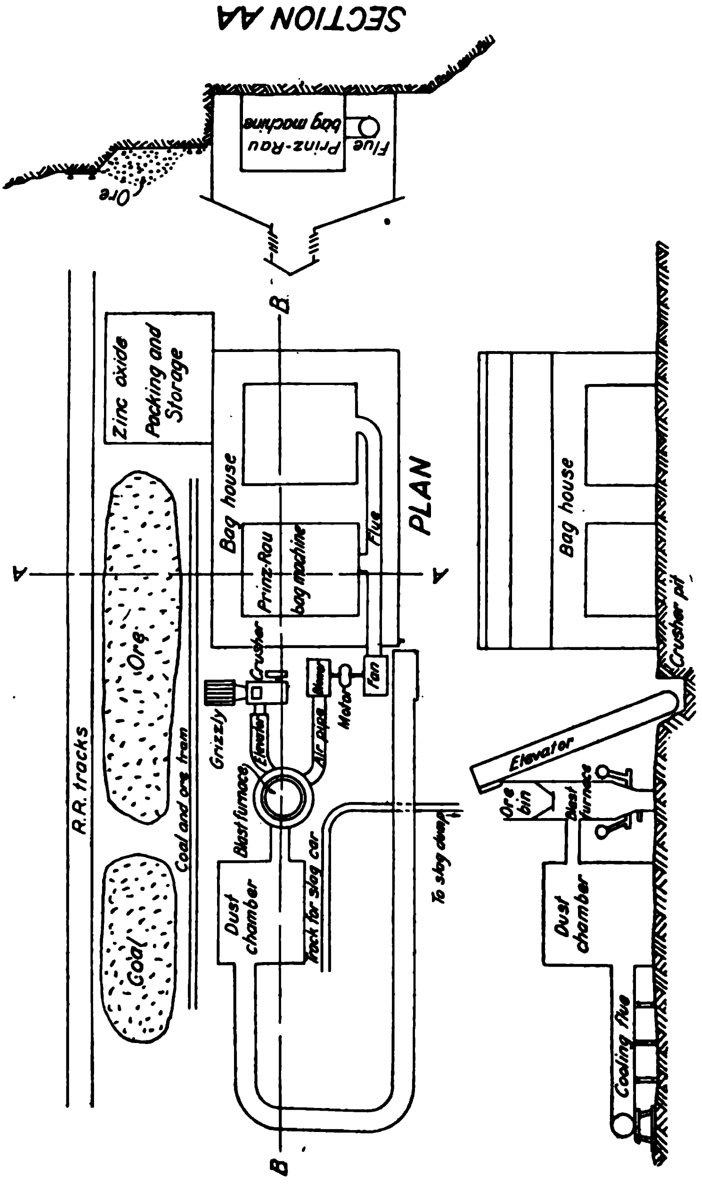

FIGURE 13.—Plan and sections of 25-ton zinc-oxide plant.

cooling pipe would radiate 0.025 B. t. u. from the gases. A rough sketch of the plant necessary (not drawn to scale) is shown in figure 13. The itemized costs of construction are as follows:

Cost of a 25-ton plant for igneous concentration of zinc ores.

One 42-inch blast furnace	$1,500
900 feet of cooling pipe, 30 inches in diameter	4,000
Two Buffalo blowers, No. 7 size	800
One 7 by 10 inch Blake crusher	500
Two Prinz Rau bag houses, No. 143 size	1,500
One brick-dust chamber	500
Supports for pipe	400
Connections to furnace	200
90-horsepower motor	1,000
Lumber	500
Buildings	1,600
Concrete	400
Supplies, etc	1,800
Labor and erection	3,000
Freight and miscellaneous	3,300
Total approximate cost	21,000

The furnace chosen could easily handle twice the estimated amount of ore by increasing the blowing equipment and the facilities for handling the slag. The crushing equipment is ample for 50 tons daily, and the only feature that would have to be altered much for increasing the plant to this capacity would be the length or size of the cooling flue and the bag house. The difference in cost for a 50-ton plant would probably not exceed $5,000. However, all the calculations have been for a 25-ton plant, in order that a rough idea could be obtained of the cost of producing a ton of zinc-oxide concentrate.

On the assumptions that only 80 per cent of the zinc will be recovered, that the product will contain 75 per cent zinc, that 25 per cent of coke on the weight of the ore will be necessary, and that lime rock can be quarried in the vicinity of the plant and delivered for $1.50 per ton, the cost per ton of ore of treatment in the 25-ton plant will be as shown in Table 39 and plotted in figures 14 and 15. The daily cost of operating the plant will be as follows:

Daily cost of operating 25-ton plant.

	Cost per day.
Depreciation, at 20 per cent	$12.00
Labor, 13 men	48.00
Power, 70 horsepower, at 2 cents per horsepower hour	35.00
Flux, cooling water, etc	8.00
Coke, 6 tons, at $8	48.00
Total metallurgical cost	151.00
Mining, at $2.50 per ton	62.50
Total, mining and metallurgical	213.50
Total cost per ton	8.55

TABLE 39.—*Costs and profits of a 25-ton zinc oxide plant.*

[Cost per ton of ore, based on 80 per cent recovery, 75 per cent zinc in product, $7.50 per ton of zinc oxide, and freight to "gas belt."]

Spelter, cents per pound.	Zinc in ore (per cent).	Metallurgical cost.	Mining cost.	Freight on ZnO.	Total cost.	On oxide contract.		On ore contract.	
						Value of ZnO.	Net profit per ton of ore.	Value of ZnO.	Net profit.
5	10	$6.05	$2.50	$0.75	$9.30	$4.90	Loss.	$5.70	Loss.
5	15	6.05	2.50	1.12	9.67	7.30	Loss.	8.55	Loss.
5	20	6.05	2.50	1.50	10.05	9.80	Loss.	11.40	$1.35
5	25	6.05	2.50	1.87	10.42	12.20	$1.78	14.25	3.83
5	30	6.05	2.50	2.25	10.80	14.60	3.80	17.10	6.30
5	35	6.05	2.50	2.62	11.17	17.10	5.93	19.95	8.78
7	10	6.05	2.50	.75	9.30	6.80	Loss.	6.50	Loss.
7	15	6.05	2.50	1.12	9.67	10.25	.58	9.75	.08
7	20	6.05	2.50	1.50	10.05	13.65	3.60	13.00	2.95
7	25	6.05	2.50	1.87	10.42	17.10	6.68	16.25	5.83
7	30	6.05	2.50	2.25	10.80	20.50	9.70	19.50	8.70
7	35	6.05	2.50	2.62	11.17	23.90	12.73	22.75	11.58
9	10	6.05	2.50	.75	9.30	8.75	Loss.	7.20	Loss.
9	15	6.05	2.50	1.12	9.67	13.15	3.48	10.80	1.13
9	20	6.05	2.50	1.50	10.05	17.50	7.45	14.40	4.35
9	25	6.05	2.50	1.87	10.42	21.90	11.48	18.00	7.58
9	30	6.05	2.50	2.25	10.80.	26.30	15.50	21.60	10.80
9	35	6.05	2.50	2.62	11.17	30.70	19.53	25.20	14.03

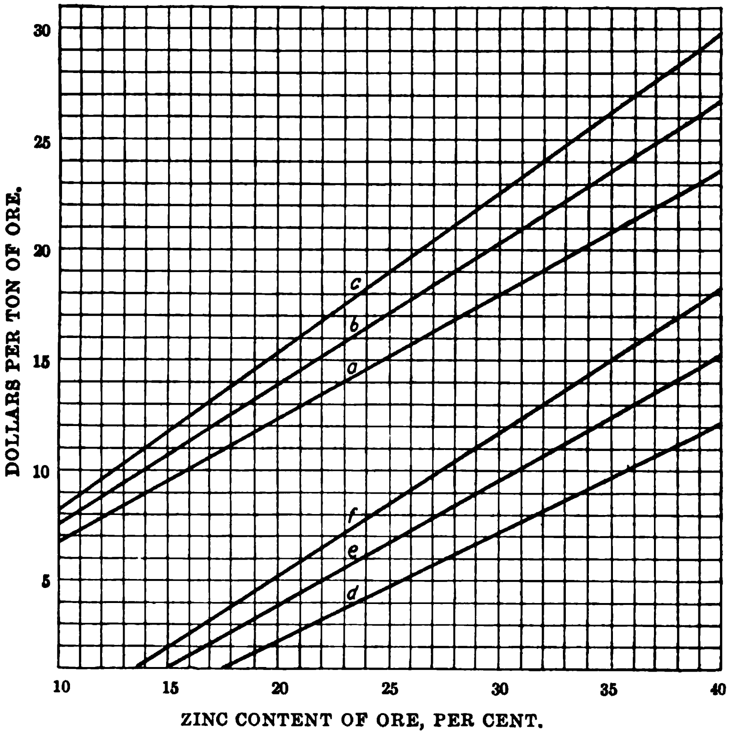

FIGURE 14.—Curves showing cost per ton of ore of operating a 25-ton igneous concentration plant, and returns as calculated from "ore" contract. Curves *a*, *b*, and *c* show value of zinc oxide product and *d*, *e*, and *f* the net profit per ton with spelter at 5, 7, and 9 cents per pound, respectively. Metallurgical cost estimated at $6.05 and mining cost at $2.50 per ton; freight on zinc oxide product variable.

For a similar plant treating 50 tons daily the expenses per ton of ore would be less. The original cost of the plant would not be much greater than for a 25-ton plant, and the amount of labor would be only slightly greater than for the 25-ton plant. Likewise the power bill would not be much larger, and the only item that is relatively fixed per ton of ore is that of fuel. This makes a probable metallurgical cost of $4.80 per ton of ore. Mining on this scale could probably be done for $2 per ton in oxidized deposits near the surface, whereas freight on the zinc-oxide product would be the same as in the former calculations. Thus the returns from a 50-ton plant would be larger, as shown in Table 40 and figures 16 and 17.

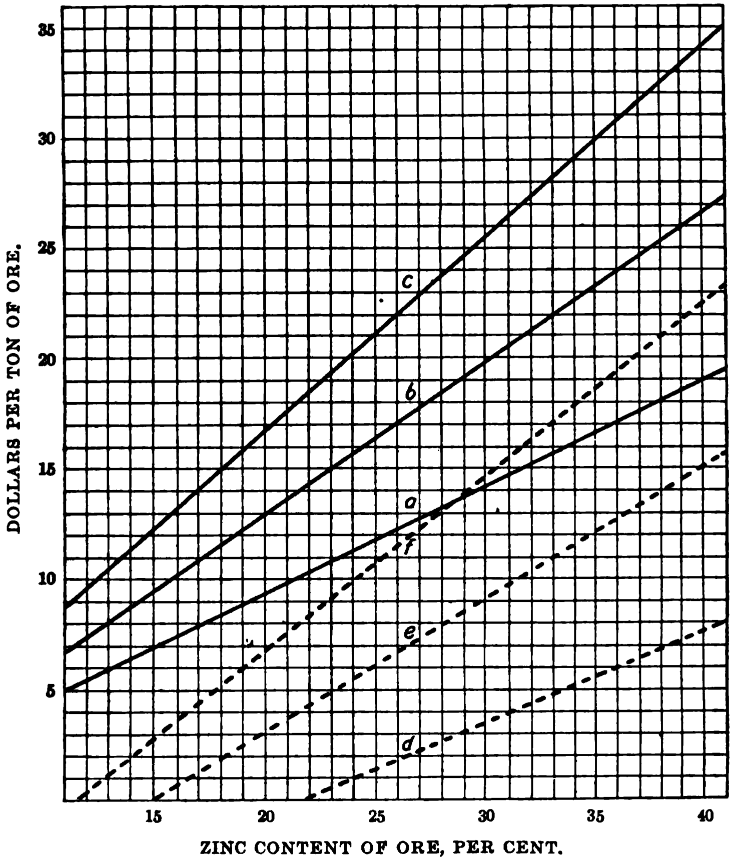

FIGURE 15.—Curves showing cost per ton of ore of operating a 25-ton igneous concentration plant, and returns as calculated from "zinc oxide" contract. Curves *a*, *b*, and *c* show value of zinc oxide product and *d*, *e*, and *f* the net profit with spelter at 5, 7, and 9 cents a pound, respectively. Metallurgical cost is estimated at $6.05 and mining cost at $2.50 per ton of ore; freight on zinc oxide product variable.

TABLE 40.—*Costs and profits of a 50-ton zinc oxide plant.*

[Cost per ton of ore, based on 80 per cent recovery, 75 per cent zinc in product, $7.50 per ton of zinc oxide, and freight to "gas belt."]

Spelter, cents per pound.	Zinc in ore, per cent.	Total cost per ton.	Oxide contract.		Ore contract.	
			Value of ZnO.	Profit per ton of ore.	Value of ZnO.	Profit per ton of ore.
5	10	$7.55	$4.90	Loss.	$5.70	Loss.
5	15	7.92	7.30	Loss.	8.55	$0.63
5	20	8.30	9.80	$1.50	11.40	3.10
5	25	8.67	12.20	3.53	11.25	5.58
5	30	9.05	14.60	5.55	17.10	8.05
5	35	9.42	17.10	7.68	19.95	10.53
7	10	7.55	6.80	Loss.	6.50	Loss.
7	15	7.92	10.25	2.33	9.75	1.83
7	20	8.30	13.65	5.35	13.00	4.70
7	25	8.67	17.10	8.43	16.25	7.58
7	30	9.05	20.50	11.45	19.50	10.45
7	35	9.42	23.90	14.48	22.75	13.33
9	10	7.55	8.75	1.20	7.20	Loss.
9	15	7.92	13.15	5.23	10.80	2.92
9	20	8.30	17.50	9.20	14.40	6.10
9	25	8.67	21.90	13.23	18.00	9.33
9	30	9.05	26.30	17.25	21.60	12.55
9	35	9.42	30.70	21.28	25.20	15.78

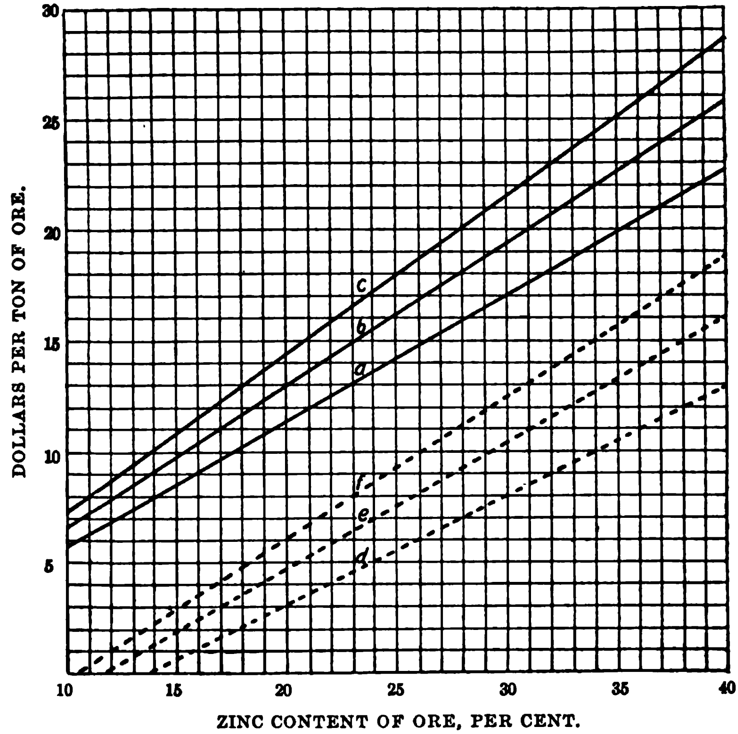

FIGURE 16.—Curves showing cost per ton of ore of operating a 50-ton igneous concentration plant, and returns based on "ore" contract. Curves *a*, *b*, and *c* show value of zinc oxide product, and *d*, *e*, and *f* the net profit with spelter at 5, 7, and 9 cents, respectively. Metallurgical cost is estimated at $6.05 and mining cost at $2.50 per ton; freight on product variable.

Both sets of calculations show that the returns depend largely on the kind of contract obtained for the sale of the zinc oxide.

By comparing the curves for the value of the crude ore, as measured by the smelter contract, with those for the returns possible from igneous concentration, it can be seen that ore of about 35 per cent zinc content brings about as much per ton either way, but that all lower grades of oxidized zinc ores (in the zinc districts of Utah or similarly situated localities) can be made to yield a greater profit by igneous concentration. This was the conclusion sought in doing the work described.

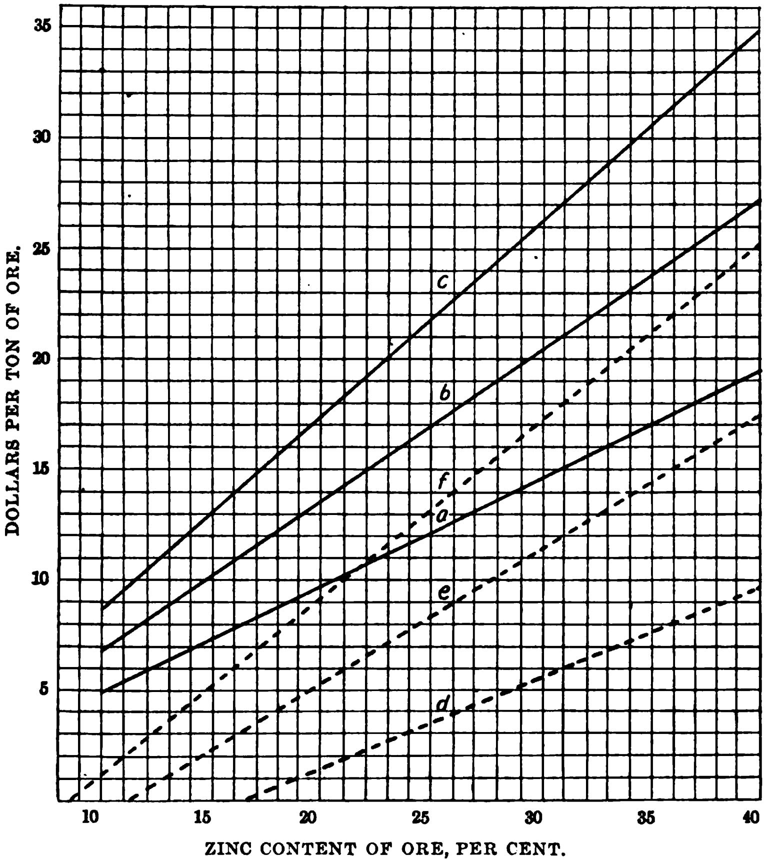

FIGURE 17.—Curves showing cost per ton of ore of operating a 50-ton igneous concentration plant, and returns as calculated from "zinc oxide" contract. Curves *a*, *b*, and *c* show value of zinc oxide product and *d*, *e*, and *f* the net profit with spelter at 5, 7, and 9 cents, respectively. Metallurgical cost is estimated at $6.05 and mining cost at $2.50 per ton of ore; freight on product variable.

SPECIAL ROASTING PROCESSES FOR OXIDIZED ZINC ORES.[a]

At the Salt Lake City station several laboratory experiments have been made in the attempt to find some cheap reagent for obtaining soluble zinc compounds from oxidized ores of zinc without much success.

Chloridizing roasts with sodium chloride showed that it was almost impossible to chloridize all the zinc to soluble form at temperatures ranging from 500° C. to as high as 1,000° C. Both oxidizing and reducing atmospheres and reverberatory furnaces as well as blast roasters, were used.

In blast roasting not more than 25 per cent of the total zinc in the ore could be consistently either chloridized and leached, or volatilized, although erratic results giving recoveries as high as 50 per cent of the zinc were obtained. The presence of pyrite or other sulphides to provide sulphur for aiding the chloridizing reactions seemed to improve matters very little. So far as the authors know there is no easy way of consistently chloridizing oxidixed zinc ores with sodium chloride or with combinations of sodium chloride and pyrite in any type of roaster.

The mixing of sulphide ores of zinc with oxidized ores, in the hope that the zinc could be sulphated by slow blast roasting, also gave low results, as not more than a third of the zinc could be made water-soluble.

Chloridizing volatilization involves mixing the ore with sodium chloride, or other chlorides, and heating to a high enough temperature to drive off fumes of zinc chloride. Reverberatory or kiln roasters must be used. Results with this method were erratic, although higher recoveries could be obtained than were possible in the blast roaster. As much as 75 per cent of the zinc in the ore could be consistently recovered if special precautions were taken. The presence of sulphides, such as pyrite, evidently had no beneficial effect on the reaction, but the addition of manganese dioxide and the sulphates of sodium and calcium seemed to be aids in maintaining good recoveries. In almost none of the tests was more than 75 per cent of the zinc obtained.

Other roasting processes, incidental to some of the leaching tests reported in subsequent chapters, are discussed in that connection.

HYDROMETALLURGY OF OXIDIZED ZINC ORES.[b]

The oxidized ores of zinc have often been regarded as being the most desirable zinc ores for hydrometallurgical treatment because they contain the zinc in an easily soluble form. The zinc in oxidized ores is usually present as either the carbonate or the silicate, both of which are easily soluble in weak acids. Hence many people, especially patentees, have claimed that the hydrometallurgical treatment of such ores presents no especial problem. The experience of the writers has been to the contrary and the number of difficulties encountered in the treatment of these ores has been somewhat surprising.

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[a] Experimenter: C. L. Larson.

[b] Experimenters: J. F. Cullen, W. G. Woolf, A. E. Gartside, H. C. Neeld, O. E. Young, C. W. Frith, R. J. Hinman, and O. C. Ralston.

Most of the western oxidized zinc ores are replacement deposits in dolomite or limestone, and as a rule consume large quantities of acid when acid leaching is attempted. Furthermore, such ores nearly always contain a considerable amount of zinc silicate, which forms silicic acid when the ore is leached with weak acid solutions. This silicic acid turns to jelly when the solutions are neutralized, causing difficulty in the filtration of the ore pulp.

At the Salt Lake City station leaching tests were made with sulphuric, hydrochloric, sulphurous, and hydrofluosilicic acids. In every test the lime, magnesia, and other carbonates accompanying the zinc caused high acid consumptions. On that account alkaline leaching solutions were tried; also caustic soda and ammoniacal solutions of various kinds.

ACID EFFICIENCY.

When an ore is treated with an acid for the purpose of recovering a metal such as zinc, the acid can not be regarded as doing useful work if it reacts with any other metals present. Hence the "efficiency" of the acid is the percentage of the total quantity of acid consumed that is actually used in dissolving the metal sought. The object of the first experiments was to compare the properties of sulphuric and hydrochloric acids in leaching a mixed zinc-lime carbonate ore. It was thought that a dilute solution of sulphuric acid might tend to coat the calcium carbonate particles with calcium sulphate, so that the acid would have a better chance to attack the zinc minerals and thus be more efficient than hydrochloric acid.

The ore tested was from the May Day mine, of Eureka, Utah, and was numbered May Day No. 2. An analysis is shown in Table 3 (p. 21). This ore contained 12.96 per cent CaO and 15.7 per cent Zn, with practically no other acid-soluble constituents, except a small proportion of hydrated iron oxides. It was found that practically no iron went into solution until the zinc and calcium minerals had reacted with the acid. Several series of leaches were made with this ore.

LEACHES WITH WEAK ACID SOLUTIONS.

In the first series successive additions of acid were made in order to leach the zinc in successive fractions. The efficiency of the acid and the percentage extraction of the zinc were determined after each addition of acid had had time to react. In each test the ore had been ground to pass a 150-mesh screen, and was leached for 16 hours to insure complete reaction. The ore and the dilute acid were agitated in small bottles on a roller-agitator designed for laboratory testing.

The results are shown by the curves in figures 18 and 19. The curves indicate that sulphuric acid has a higher efficiency, but the hydrochloric acid gives a higher recovery of the zinc, pound for pound of acid used. As hydrochloric acid has a lower molecular weight than sulphuric acid, a pound of the former should theoretically do more work than a pound of the latter, which is in agreement with the observed results. If the curves for percentage extraction of zinc are plotted with moles (molecular weights) of acid in place of pounds of acid per ton of ore, it is found that the work done by the two acids is identical—that one molecule of hydrochloric acid does exactly the same amount of chemical work in leaching zinc as does one molecule of sulphuric acid. (See fig. 20). Hence, chemically, the two acids are equally efficient as regards the percentage recovery of the zinc.

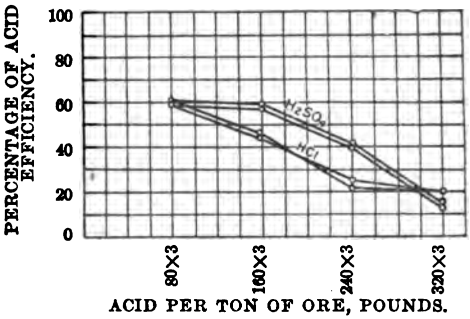

FIGURE 18.—Curves showing efficiency of weak acids in four successive leaches.

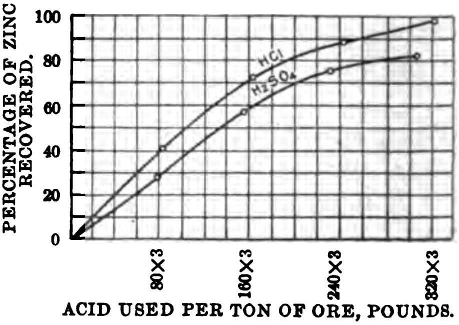

FIGURE 19.—Curves showing recovery of zinc in four successive leaches with weak acids (80 pounds of acid per ton of heads used in each leach).

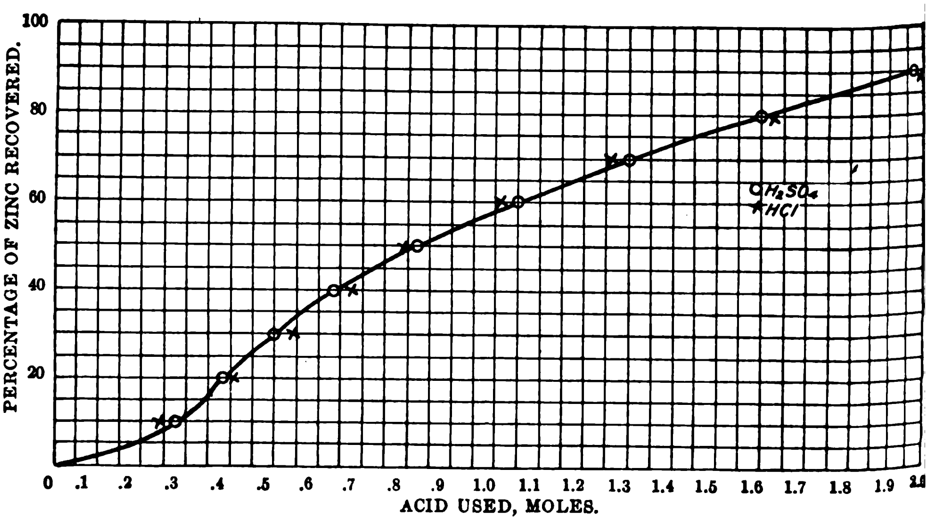

FIGURE 20.—Curve showing relation of zinc recovery to molecular weights of acid used.

LEACHES WITH INCREASING STRENGTHS OF ACID SOLUTION.

In addition to the series of leaches with successive amounts of weak acid, a series of leaches were made with the same weights of acid but with increasing concentrations, in order that an idea might be obtained as to the effect of different concentrations of acid on the leaching of the zinc and on the acid efficiency of the two acids. The results are plotted in figures 21 and 22, which also show similar curves for the first series. Comparison of these curves show that absolutely no difference in chemical effect is produced by the use of acid in various concentrations. In other words, a definite quantity of acid, when applied to a certain weight of ore and permitted sufficient time to react, will leach out a certain amount of zinc with a certain definite efficiency of the acid, whatever the concentration of the acid solution.

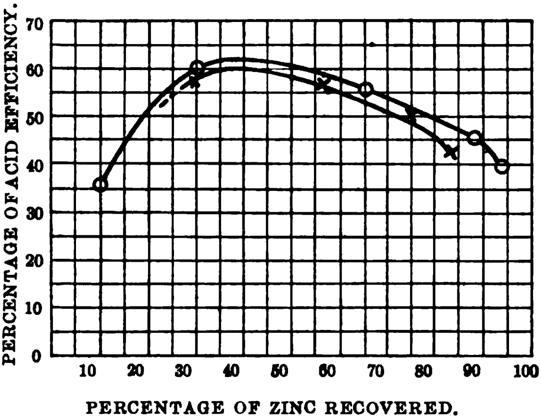

FIGURE 21.—Curves showing effect of different concentrations of sulphuric acid: X, successive leaches with weak acid; O, leaches with different strenghs of acid solution.

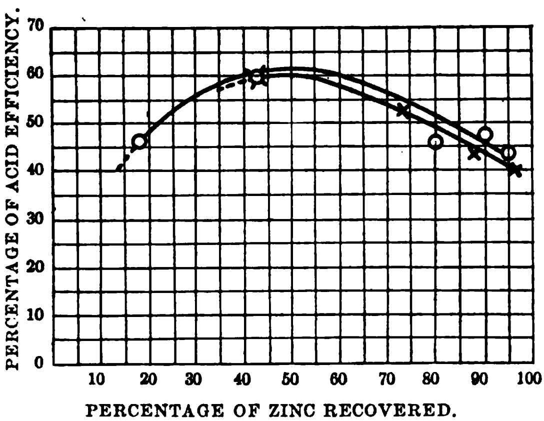

FIGURE 22.—Curves showing effects of different concentrations of hydrochloric acid: X, successive leaches with weak acid; O, leaches with different strengths of acid solution.

These results are of fundamental importance, because they show that any convenient concentration of acid may be used, the exact concentration being entirely determined by mechanical considerations in designing the leaching plant. Subsequent work on a large variety of ores verified this result, so that the data need not be presented.

Curves for computing the acid efficiencies of sulphuric and hydrochloric acids when the number of pounds of zinc leached from a sample of ore and the number of pounds of acid consumed are known, are presented in figure 23. These curves can likewise be used in interpreting the results of some of the tests described in the following pages, when the acid efficiencies are stated, for finding the weight of acid necessary to recover a pound of zinc from the ore in question.

LEACHING AGENTS.

SULPHURIC ACID.

As the foregoing tests had indicated that the strength of acid solution used had no effect either on the acid efficiency or on the percentage of zinc recovered, further tests were made of a wide variety of oxidized zinc ores in order to confirm these conclusions and to study the various impurities that might be found in the solutions from different ores.

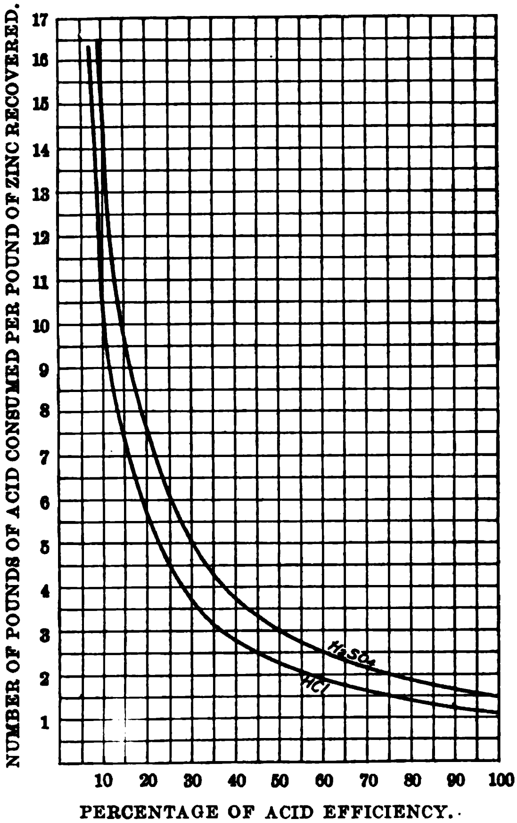

FIGURE 23.—Curves for computing acid efficiency when weights of acids consumed and zinc recovered are known.

The first series of leaches was with samples representing different types of oxidized ores of zinc from various mining districts. The composition and the screen analysis of each of these samples are shown in Table 41, following:

TABLE 41.—*Compositions of oxidized zinc ores used in leaching tests with sulphuric acid.*

	Scranton No. 1.	Scranton No. 2.	Scranton No. 3.	Lower Mammoth No. 1.	Lower Mammoth No. 2.	Lower Mammoth No. 3.	Bullion Coalition No. 13.	Western Mining Co.	Santaquin King mine.	I ake View No.103.	May Day No. 4.
Insoluble....per cent..	14.0	10.6	8.5	54.6	59.6	19.2	17.0	19.5	33.9	14.0	25.0
CaO.............do....	13.1	8.2	12.4	2.9	.9	2.1	4.2	Trace.	.6	19.5	12.2
Fe...............do....	7.5	33.0	18.5	10.5	4.79	27.0	18.5	19.7	.3	.8	2.6
Al_2O_3..........do....	11.3	3.3	2.6	2.3	4.48	12.1	6.1	3.3	Trace.	1.5	....
MgO.............do....	1.5	4.7	1.1	1.4	.4	2.1	1.5	....	..do..	Trace.	3.9
Zn...............do....	25.0	8.9	18.7	9.1	12.3	11.3	28.3	20.1	45.5	34.3	21.6
Pb...............do....	.3	1.4	.5	1.2	.3	2.2	.3	.05	Trace.	1.4	11.7
Cu...............do....	Trace.	Trace.	Trace.	Trace.	Trace.	.99	Trace.	....	..do..	....	Trace.
Ag............ounces..	.45	.28	.16	1.8	.3	4.3	....	....	....	....	....
S............per cent..	.3	.35	.5	4.8	1.04	.9	.4	....	....	....	.36
CO_2.............do....	16.8	15.5	17.6	3.6	8.5	6.1	11.9	....	....	....	17.1
Mn..............do....	.04	.05	.19	.53	.27	.72	.63	....	....	.03	1.07

SCREEN ANALYSIS IN CUMULATIVE PERCENTAGES.

Mesh.	Scranton No. 1.	Scranton No. 2.	Scranton No. 3.	Lower Mammoth No. 1.	Lower Mammoth No. 2.	Lower Mammoth No. 3.	Bullion Coalition No. 13.	Western Mining Co.	Santaquin King mine.	I ake View No.103.	May Day No. 4.
+28..................	8.5	6.9	11.4	14.3	12.0	10.5	2.6	11.2	(a)	8.6	11.4
35....................	20.8	29.2	25.6	22.8	22.1	22.8	7.1	26.1	(a)	25.7	27.8
48....................	32.9	41.8	37.9	28.8	30.4	34.4	15.3	36.9	(a)	38.4	40.4
65....................	43.5	52.0	47.3	33.2	36.3	44.2	28.2	47.2	(a)	49.8	50.1
100..................	56.2	65.7	58.1	39.2	43.0	57.1	39.9	57.9	(a)	61.0	61.4
150..................	63.2	72.8	64.1	43.2	47.8	66.7	47.4	63.8	(a)	69.4	69.1
200..................	74.2	84.7	73.8	46.8	51.7	74.3	74.6	74.2	(a)	73.0	74.1
	100	100	100	100	100	100	100	100	(a)	100	100

[a] All passed 60 mesh.

Most of these ores were tested with dilute acetic acid to dissolve zinc carbonate and with dilute sulphuric acid to dissolve zinc carbonate and zinc silicate. The results of the tests, shown in Table 42, give a rough idea of the total percentage of oxidized zinc present as carbonate.

TABLE 42.—*Results of trial leaches on 1-gram samples of oxidized ores.*

[Samples ground to pass through 100-mesh sieve.]

Mine and sample.	Percentage of zinc recovered by leaching for—		
	20 minutes in hot acetic acid.	30 minutes in hot sulphuric acid solution, 10 per cent strength.	90 minutes in cold sulphuric acid solution, 10 per cent strength.
Scranton, No. 1..	61.5	95.1	90.0
Scranton, No. 2..	69.7	96.0	89.0
Scranton, No. 3..	64.3	94.0	85.7
Lower Mammoth, No. 1..	49.7	80.5	56.8
Lower Mammoth, No. 2..	67.4	94.0	75.6
Lower Mammoth, No. 3..	72.5	87.3	93.0
May Day, No. 4..	89.8	97.0	91.2
Bullion Coalition..	84.5	93.5	95.5

The ores were then tested for acid consumption and zinc recovery by treating 100-gram lots, ground to pass a 20-mesh sieve, in sulphuric acid solutions of about 10 per cent strength. The ore and solution was placed in 2.5-liter acid bottles and agitated on a set of agitating rolls. The principal results of each set of tests are reported in Table 43 following. Most of the zinc can be removed from the average oxidized zinc ore by the use of sulphuric acid solutions, but the efficiency of the acid used in leaching varies greatly for different ores, being naturally dependent on the amounts of lime, magnesia, and other acid-consuming constituents of the ore.

TABLE 43.—*Results of leaching oxidized zinc ores with sulphuric acid, showing zinc recovery and acid efficiency.*

[100-gram sample, ground to pass 20-mesh sieve, used in each test.]

Mine and sample.	Time, hours.	Equivalents of acid supplied.	Percentage of acid efficiency.	Percentage of zinc recovered.
Scranton, No. 1..........	1.5	1.0	53	53
	1.5	1.5	60	90
	1.5	2.0	60	95
	3.0	1.0	61	92
	3.0	1.5	61	91
	3.0	2.0	59	90
Scranton, No. 2..........	1.5	1.5	45	87
	1.5	2.36	35	81
	1.5	4.45	36	80
	3.0	2.36	34	81
	3.0	4.45	34	82
Scranton, No. 3..........	1.5	0.69	54	54
	1.5	1.0	41	84
	1.5	1.3	45	87
	3.0	1.0	43	87
	3.0	1.3	46	89
Bullion Coalition, No. 15.	1.5	1.0	85	85
	1.5	1.5	80	92
	1.5	2.0	80	93
	3.0	1.5	78	92
	3.0	2.0	79	93
May Day, No. 4.........	1.5	1.0	86	66
	1.5	1.5	63	86
	1.5	2.0	67	90
	3.0	1.5	64	92
	3.0	2.0	68	90
Lower Mammoth, No. 1.	1.5	1.0	83	52
	1.5	3.1	85	52
	1.5	4.6	85	52
	3.0	3.1	85	52
	3.0	4.6	85	52
Lower Mammoth, No. 2..	1.5	1.5	82	75
	1.5	3.1	80	75
	1.5	4.3	80	77
	3.0	3.1	82	77
	3.0	4.3	81	75
Lower Mammoth, No. 3..	1.5	1.5	53	80
	1.5	3.5	52	84
	1.5	4.7	52	82
	3.0	3.5	48	84
	3.0	4.7	48	84
Santaquin King..........	1.5	1.0	82.5	79
	1.5	1.25	81.0	88
	1.5	1.8	82	90
	3.0	1.25	93	94
	3.0	1.8	94	96
Lake View, No. 105.......	1.5	1.0	80	75
	1.5	1.5	67	96
	1.5	2.4	60	97
	3.0	1.5	66	97
	3.0	2.4	60	98
Western Mining Co.......	1.5	1.0	72	80
	1.5	1.25	80	90
	1.5	2.5	85	91
	3.0	1.25	84	92
	3.0	2.5	86	93

Such tests are of fundamental importance in determining whether enough zinc can be recovered from a definite ore and whether the consumption of acid will be excessive, but an even more important point is the properties of the solutions made, because the purity of the finished product depends on the character of the solutions. As the modern methods of purifying zinc sulphate solutions depend upon reducing the solution to a neutral condition, analysis of the solutions from such ores, when only sufficient acid is used to dissolve the zinc and to leave the solution neutral, is of prime importance. On that account, a sample of each of the ores was leached with sulphuric acid solution of 10 per cent strength for 1½ hours. Then enough concentrated acid was added to bring the acidity back to 10 per cent, and the solution was passed over fresh ore until no free acid remained. The analyses of the resulting solutions are shown in Table 44. In almost every test it was noted that when the pulp was neutral there was some difficulty in filtering the solution. Neutral solutions from Scranton No. 2 and No. 3 ores were almost impossible to filter and finally set as gels, on account of the high percentage of silicic acid present in them. Some of the gel that formed in one test was washed and dried, and gave the following analysis: Zn, 2 per cent; Fe, 2 per cent; CaO, 0.7 per cent, and SiO_2, 93.6 per cent.

TABLE 44.—*Results of analyses of neutral solution of zinc sulphate and impurities.*

Mine and sample.	Zn.	SiO_2.	CaO.	MgO.	Al_2O_3.	Fe.	Mn.	Zn in tailing.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Scranton, No. 1	9.3	0.46	0.28	0.72	0.01	0.03	....	1.4
Scranton, No. 2	6.2	2.05	.23	1.2	....	.07	....	1.7
Scranton, No. 3	7.0	2.2	.20	.75	....	.02	0.008	1.5
Bullion Coalition, No. 13	12.7	1.76	.27	.20	....	.07	.0008	3.7
Lower Mammoth, No. 1	8.75	.07	.16	.34	.22	.40	....	4.4
Lower Mammoth, No. 2	8.20	.39	.24	.22	.19	.17	.003	4.1
Lower Mammoth, No. 3	8.0	.22	.19	.26	.90	.08	....	2.6
May Day, No. 4	9.75	.14	.19	.26	....	.02	.015	1.7
Santaquin King	12.6	.33	.2	tr.	.27	.02	....	1.6
Western Mining	14.4	.51	.1	tr.	.53	.08	....	2.0
Lake View, No. 105	14.5	.51	.2	tr.	.52	.08	....	.5

As it was thought probable that electrolytic recovery of the zinc from such solutions might be attempted in some instances, the effects of silicic acid gel were further investigated. It was found that silicic acid caused serious trouble only when the solutions were neutral, and that the acid solutions were much more easily filtered. Many of the pulps, when allowed to stand over-night in the neutral condition could not be filtered the next morning and in many cases the whole solution has set to a gel. The lowest percentage of silica ever observed in one of these gels was 0.6 per cent and this gel took about two days to set. Electrolytic precipitation of the zinc would not remove the silica from the solutions, therefore the silica would tend to build up. The analyses of the solutions indicated that with continued use the silicic acid would soon build up to a point where it would entirely prevent further operation. Cyclic leaching of a number of the ores tested proved that the silica had such an effect. As a rule when the silica content of the solution reached more than 2 per cent, filtering became very difficult and could be done only when the neutral liquor was acidified and used for treating fresh ore without removal of the zinc.

As is shown subsequently, removal of this colloidal silica from solutions contaminated with it was almost impossible. Therefore, in treating oxidized zinc ores containing silica, the preparation of zinc sulphate solutions in such a way that the silica would not be present is necessary. Silicic acid can be dessicated to form "insoluble silica" and water by heating it to more than 125° C. in the open air. Consequently, an attempt was made to prepare a solution free of silicic acid by treating the ore with just enough strong sulphuric acid to moisten it, heating the mixture of sulphuric acid and ore to about 150° C., or higher, and leaching the calcined material with water.

It was found that concentrated sulphuric acid would not moisten the ore sufficiently, so the necessary acid for converting the zinc into zinc sulphate was added as a 50 per cent solution, with satisfactory results. The laboratory roasts were made in roasting dishes in a gas muffle furnace. A large proportion of the iron and the alumina are converted into the corresponding sulphates by this treatment, unless the temperature of the roast is maintained at about 600° C. or higher. Iron sulphate breaks up rather slowly at 600° C., requiring at least two hours' roasting to prevent the subsequent appearance of iron in the solutions. At 700° C. the necessary time for obtaining the same result varies from one-half to one hour. At 800° C. some of the zinc sulphate is broken up by the heat, and as a result leaching with water does not obtain as much zinc. The zinc sulphate formed tends to cake the material together, and in most furnaces this caking would probably cause mechanical difficulty in handling such material. Possibly some type of horizontal kiln would be necessary in order to avoid trouble from this source.

Practically all the oxidized ores tested yielded to this treatment, the zinc recoveries being about the same as those obtained by direct leaching with solutions of sulphuric acid, except that when a charge was overheated some of the zinc sulphate was broken down into zinc oxide or into basic zinc sulphate. Under these conditions sulphuric acid had to be added to the leaching water in order to get good recoveries. In almost every test it was necessary to use enough sulphuric acid to satisfy all the zinc, lead, lime, and magnesia in the ore before satisfactory recovery of the zinc could be made. The amounts of iron usually entering the solution varied from 10 to 20 milligrams per liter. Manganese was not so easy to keep out of the solution, because its sulphate does not seem to decompose as easily as that of iron. Consequently there was usually a considerable amount of manganese in the solutions prepared from these ores. Silica was rarely present in more than traces.

The disadvantage of getting the zinc into solution in this manner is that only strong solutions of sulphuric acid can be used, whereas electrolytic precipitation of a zinc sulphate solution would regenerate only a weak solution of sulphuric acid which could not be used for sulphating the ore. Consequently this method of preparing pure solutions involves chemical precipitation of the zinc from the solution and loss of the sulphuric acid. Such a procedure would naturally be limited to localities where sulphuric acid was cheaply available in large amounts.

Niter cake, the impure bisulphate of sodium ($NaHSO_4$), which is usually available as a waste product from nitric acid plants, powder plants, sulphuric acid plants, etc., has an available acidity of about 30 per cent in the usual commercial article, and melts at a low temperature. Niter cake was tried as a substitute for sulphuric acid in sulphating ores at high temperatures. The efficiency in recovering zinc was about the same as with sulphuric acid, but the sodium and zinc sulphates resulting caked the ore harder than when only sulphuric acid was used. Furthermore, with some ores the amount of niter cake necessary to satisfy the acid requirements of the ore was excessive, and the resulting burden of sodium sulphate in the water leaches made recovery of the zinc from the solution doubtful. Therefore, sodium sulphate is an undesirable diluent of the acid used in preparing the ore.

HYDROCHLORIC ACID.

The preceding considerations of the adaptability of sulphating at a high temperature followed by chemical precipitation for recovering zinc recalls the fact that no acceptable method of precipitating zinc from its sulphate solutions by any commercially available chemicals has been devised. Lime is the cheapest base available, but it forms calcium sulphate which will mix with the zinc hydroxide precipitate. Hence the preparation of chloride solutions of zinc is desirable, if this can be done, because they would be better adapted to precipitation by lime. As was shown in an earlier part of this paper, hydrochloric acid is equally efficient with sulphuric acid, by molecular weights, in dissolving zinc from its oxidized ores, and wherever cheaply available could easily be used for leaching such ores. The station has contemplated making some experiments on the preparation of hydrochloric acid by the interaction of sodium chloride with silica and steam at high temperatures. The reaction is known to take place, but development work needs to be done on the mechanical application of the reactions. If hydrochloric acid can be prepared cheaply in this way, application to the leaching of zinc ores will then be easy.

SULPHUROUS ACID.

The use of sulphur dioxide as a leaching agent has always appealed to the hydrometallurgist, but to date there have been practically no successful commercial installations of sulphurous-acid leaching plants for either zinc or copper ores. Many attempts have been made to absorb sulphur dioxide with water or steam and use the resulting sulphurous acid for leaching either oxidized zinc or copper minerals, but the mechanical difficulties of applying the process have caused all such enterprises to fail. The asphyxiating character of the fumes rising from the solutions necessitate the use of closed vessels, whereas difficulties from reprecipitation and crystallization of salts in pipes and other closed vessels have made it almost necessary to work with open vessels. Furthermore, some of the proposed methods have looked toward recovery of the sulphur dioxide from the sulphite solutions, whereas practice has proved that much of the zinc sulphite is oxidized to higher oxides, from which it was impossible to recover sulphur dioxide.

In most of the intermountain districts the recovery of sulphur dioxide is not vitally necessary, and any satisfactory method for the leaching of the zinc or the copper and the ultimate wasting of the sulphur dioxide would usually be acceptable. On that account further investigation of the possibility of sulphurous acid leaching of oxidized zinc ores was deemed advisable.

The first factor determined was the approximate solubility of zinc bisulphite, the soluble compound of zinc formed by the use of excess of sulphurous acid. The monosulphite of zinc is insoluble, and most of the proposed processes for sulphurous acid leaching involve leaching of the zinc as a bisulphite solution with subsequent precipitation of insoluble zinc monosulphite. By suspending an excess of zinc oxide in water and then passing sulphur dioxide from a cylinder of the liquified gas until the solution smelled strongly of excess sulphur dioxide, it was found that at the ordinary temperature of the laboratory 6.97 per cent of zinc was in solution and the total sulphites present were equivalent to 16.08 per cent SO_2. Thus about 2.36 per cent of SO_2 was uncombined. Most of the values in the literature on the subject for solubility of zinc sulphite agree reasonably well with these figures, although in some patent specifications mention is made that some of the zinc often oxidizes to the sulphate form, when the total zinc content of the sulphite leaching solution can approach 10 per cent.

A number of tests were made with several ores, to determine the leaching efficiency of sulphurous acid as compared with sulphuric acid. The results of these leaches are given in Table 45. The zinc recoveries, even with very dilute solutions, are fully as good as those possible with sulphuric acid solutions. The acid efficiency obtainable is likewise comparable with the acid efficiency of the sulphuric acid solutions. The figures in the column for acid efficiency represent the apparent efficiency, which is twice the actual efficiency, because in titrating the leach solution for acidity the zinc bisulphite reacts with the acid until completely neutralized, forming zinc monosulphite. Hence the figures in the column of acid efficiencies should be divided by two.

TABLE 45.—*Results of sulphurous acid leaching of oxidized zinc ores.*

Mine and sample.	Leach No.	Weight of ore.	Weight of SO_2.	SO_2 per 100 c. c. of water.	Volume of solution.	Zinc per gram of SO_2.	Percentage of acid efficiency.	Percentage of zinc recovered.
		Grams.	*Grams.*	*Grams.*	*C. C.*	*Grams.*		
Bullion Coalition, 13....	1	6.29	3.35	0.167	2,000	0.973	192.0	96.1
	2	6.29	3.35	.335	1,000	.854	168.0	96.1
	3	6.29	3.35	.670	500	.830	162.0	97.1
	4	6.29	3.35	1.34	250	.912	187.0	92.3
	5	6.29	3.35	2.00	167	.594	137.0	91.0
	6	6.29	6.70	2.68	250	.934	180.0	97.9
Scranton, 1............	1	6.36	5.00	.25	2,000	.779	192.5	95.2
	2	6.36	5.00	.50	1,000	.940	185.0	94.0
	3	6.36	5.00	1.00	500	.856	168.0	95.2
	4	6.36	5.00	2.00	250	.865	176.5	97.0
	5	6.36	5.00	3.00	167	.624	122.7	92.8

The impurities taken into solution by sulphite solutions are lime, alumina, iron, copper, and small amounts of silica. Lead, silver, and gold do not dissolve to any extent.

The mechanical requirements for successful leaching of zinc ores by the bisulphite method have already been considered in the part of this bulletin dealing with the hydrometallurgy of the sulphide ores (pp. 81–84). Except for the tendency of silicic acid to enter the solutions in the leaching of oxidized ores and later contaminate the precipitate of zinc monosulphite, there is little difference between the problem of the treatment of oxidized and of sulphide ores by sulphurous acid. As the zinc bisulphite solutions must be acid for filtering, no filtration difficulty is experienced, whereas zinc sulphate solutions are usually filtered in the neutral condition where the silicic acid causes trouble.

HYDROFLUOSILICIC ACID.

As the writers had been informed that zinc fluosilicate solutions could be easily electrolyzed to give good cathode spelter, a few hydrofluosilicic acid leaches of oxidized zinc ores were tried in order to see whether the acid would be an acceptable leaching agent. Leaching the raw ore with hydrofluosilicic acid gave an acid efficiency of more than 100 per cent. The theoretical weight of acid which should be consumed is 2.2 pounds per pound of zinc, whereas the observed consumption was only 1.9 pounds. This lower rate was attributed to the presence of some hydrofluoric acid in the hydrofluosilicic acid, which, on account of the lower molecular weight of the former, would make the hydrofluosilicic acid appear more than 100 per cent efficient.

On leaching part of the same sample of ore with the acid after calcination, the acid consumption amounted to only 0.52 pound per pound of zinc. On the assumption that the zinc oxide from the roasting caused hydrolysis of the hydrofluosilicic acid with resultant formation of zinc fluoride, in place of zinc fluosilicate, only 0.615 pound of the acid would be required to leach 1 pound of zinc. Thus, the results would seem to indicate that the following reactions took place in the two tests:

$$H_2SiF_6+ZnCO_3=ZnSiF_6+H_2O+CO_2$$
$$H_2SiF_6+3ZnO=3ZnF_2+H_2O+SiO_2$$

This shows that hydrofluosilicic acid might be used for the leaching of zinc carbonate ores, but that it would be unsuitable for the treatment of any type of roasted products. No further work was done with it.

ALKALINE AGENTS.

The fact that zinc oxide and zinc carbonate will combine with solutions of caustic soda or of ammonia to form zincates is the basis of many proposed processes for leaching oxidized zinc ores. In many instances these processes would have an especial advantage over acid leaching, because most of the oxidized zinc ores contain important amounts of calcium and magnesium carbonates, and other acid consuming minerals which would not affect the alkaline leaching agents. Hence much better chemical efficiency of the leaching agent should be possible. With this in mind, two leaching agents were tested—caustic soda solutions and ammoniacal solutions.

CAUSTIC SODA.

Caustic soda, as a solvent for zinc ores, has been proposed by many inventors, but has never been applied commercially with any success.[a] The available literature gives little information as to the causes for the failure of caustic soda as a leaching agent. However, the writers soon found that, although many patents claim the application of caustic soda solutions to oxidized ores, satisfactory recoveries of the zinc from these ores by the use of solutions containing 10 to 25 per cent NaOH was almost impossible.

The results of six leaches of the May Day No. 2 ore with caustic soda solutions are recorded in Table 46. In each of these leaches

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[a] A list of the principal United States patents covering the use of caustic soda leaching solutions is as follows: Ketchum, E. C., No. 592055, Nov. 21, 1896; Ryan, Thomas, jr., and Hughes, Newton, No. 647989, Dec. 26, 1899; Strzoda, Wilhelm, No. 656305, Jan. 20, 1899; Sadtler, S. S., No. 700563, July 10, 1900; Barton, Thomas, and McGlue, T. B., 689,275, July 30, 1901; Ranson, Charles, No. 1023964, July 8, 1910; Bretherton, S. E., No. 1204843, Dec. 23, 1912; Asef, Waldemar, No. 1165743, Oct. 26, 1914; Snyder, E. H., No. 1184585, Mar. 31, 1915.

33.3 grams of ore ground to pass an 80-mesh sieve was agitated in 100 cc. of a 22.44 per cent NaOH solution. With cold solutions the recovery was very low. Even with hot solutions only about two-thirds of the zinc was obtained and the solution had to be maintained at 97°C. for four hours for this result. The zinc oxide is supposed to be precipitated from the solution on dilution, hence enough water was added to make 1,000 cc., which, of course, reduced the strength of the solution to about 2 per cent. Under these conditions a precipitate of zinc hydroxide was obtained containing about 40 per cent of the zinc originally in the ore, as shown by the last column in Table 46. The remainder of the zinc stayed in solution.

TABLE 46.—*Results of leaching oxidized zinc ore with caustic soda solution.*

Test.	Temperature.	Time.	Tailing.		Zinc recovery.	Zinc per 100 c. c. of solution.	Zinc recovered on dilution.
			Weight.	Zinc content.			
	° C.	*Hrs.*	*Grams.*	*P. ct.*	*P. ct.*	*Grams.*	*P. ct.*
1	97	1	26.42	7.4	62.5	3.26	34.5
2	97	2	26.14	7.3	63.5	3.38	41.5
3	97	3	26.01	7.0	65.2	3.47	39.0
4	97	4	25.72	7.7	62.2	3.44	40.7
5	20	20	29.25	11.1	37.8	1.93	....
6	20	7	30.02	12.5	28.2	1.53	....

On account of the difficulty of dissolving the zinc, to say nothing of the practical difficulties involved in dilution of the solution followed by evaporation in order to recover the sodium hydroxide, no further work was done with this process.

The consumption of alkali, determined in each of the tests shown in Table 46, was about 80 to 100 pounds of sodium hydroxide per ton of ore. The sodium is supposedly fixed as sulphate or chloride, or possibly even as silicate, but the results had been so discouraging that this latter point was not determined.

AMMONIA.

Ammoniacal leaching solutions have received much more favorable consideration than caustic soda from metallurgists in times past, although no commercial plants using such solutions have been built.[a]

Most of the methods proposed for leaching zinc ores with ammonia involve the use of ammoniacal ammonium-carbonate solutions, because carbon dioxide tends to accumulate in the solutions as ammonium carbonate, and also because the presence of combined carbon dioxide seemingly does not impair the solubility of the zinc oxide and other oxidized compounds of zinc. Ammonium zincate is a compound which is often assumed to result from the reaction of the solutions on the zinc minerals. Whether the zinc oxide actually combines with the ammonia, or whether it is merely dispersed and taken into colloidal solution, is immaterial, as long as the zinc can be dissolved.

---

[a] United States patents relating to ammoniacal leaching of zinc ores are as follows: Burghardt, C. A., and Rigg, Gilbert, 585355, June 1, 1896; Koehler, W. J., 611917, Nov. 7, 1896; Howard, Henry, 623154, May 14, 1898; Rigg, Gilbert, 654804, May 6, 1899; Hirsching, Henry, 735512, Mar. 19, 1902; Bermont, Victor, 751712, May 6, 1902; Joseph, T. B., 780293, May 18, 1904.

A lengthy series of tests with solutions of ammonia and solutions of ammoniacal ammonium sulphite was made in the effort to determine under just what conditions the zinc could be most easily leached. It was found that the solutions containing ammoniacal ammonium sulphite would dissolve twice as much zinc from the ore as the ammonia solutions. A solution containing about 5 per cent free NH_3 and about 3 per cent $(NH_4)_2SO_3$ gave as good recoveries as stronger solutions. The amount of zinc recovered was, then, roughly proportional to the free ammonia present, being about 0.29 gram of zinc per gram of free NH_3. This latter ratio shows that 14 molecules of NH_3 are required to extract 1 atom of zn. Such leaching solutions dissolved slightly more zinc than would dilute acetic acid, when applied to the leaching of an oxidized zinc ore. Evidently the ammonial solutions attack only the zinc carbonate, leaving the zinc silicate minerals almost unaffected. On leaching a roasted sphalerite ore it was found that all the zinc oxide, zinc sulphate, and zinc sulphite were dissolved. Zinc sulphide and zinc ferrate were seemingly unaffected.

The large amount of ammonia necessary to dissolve the zinc makes such a process of doubtful value, because either the proportion of leaching solution to ore must be large or else strong solutions of ammonia must be used, in which event only about a 5 per cent solution of zinc can be prepared. Neither of these conditions is desirable.

Therefore, the conclusion may be made that ammoniacal leaching solutions are not adapted to complete recovery of zinc from either roasted sulphide ores or oxidized zinc ores, except certain sulphide ores that can be roasted without the formation of ferrites, or oxide ores that contain no zinc silicate. In any event the mechanical difficulties in handling the leaching solutions are serious.

PURIFICATION AND PRECIPITATION OF SOLUTIONS.

The solutions of zinc prepared from oxidized ores by various leaching processes will not differ greatly from those prepared from the sulphide ores, except in a few particulars. As a rule, a greater proportion of impurities will be taken up from the oxidized ores, hence the solutions will require more careful purification.

Silicic acid was the most prominent impurity noted in the work with oxidized ores. Acid leaching solutions attack the zinc silicate, which is almost invariably present in the oxidized ores, forming silicic acid which tended to set to a gel as soon as the solutions became neutral. As most of the iron, alumina, and arsenic are removed from the zinc sulphate or zinc chloride solutions while in neutral condition, this formation of silicic acid introduced a serious difficulty. Attempts to precipitate the silicic acid by the use of zinc oxide, lime, and glue, resulted in a light fluffy precipitate suspended in the solution which was difficult to filter and retained too much zinc. No satisfactory method of purifying the solution of silicic acid could be found. This resulted in the adoption of methods of ore treatment that would convert the zinc to a soluble form while breaking up the silicic acid by desiccation.

Limonite and similar compounds in oxidized zinc ores are readily attacked by sulphuric acid solutions, with the result that much iron is dissolved and must be removed before either satisfactory electrolytic or chemical precipitation can take place. The removal of iron from such solutions, however, is not difficult and has been discussed sufficiently in the part of this bulletin on the hydrometallurgy of zinc sulphide ore. The same statement applies to the manganese in these ores.

The precipitation of the various leaching solutions has been hinted at during the discussion of the leaching. Precipitation of these solutions is no different from the precipitation of the solutions prepared from sulphide ores, hence need not be discussed at this point except where the treatment of oxidized ores may introduce some new factor not previously mentioned.

There are no commercial plants in operation at present for treating oxidized zinc ores by leaching and precipitation. As sulphuric acid is practically the only leaching agent used in the United States, and electrolytic precipitation is the only method in common use, seemingly some plant for treating oxidized zinc ores by a combination of these methods would have been established if the method were feasible. The writer's experience in leaching raw oxidized zinc ores has been that the difficulty of removing the silicic acid makes purification of the solutions for satisfactory electrolytic precipitation impossible. The only feasible course was to treat the ore with strong sulphuric acid solutions, and heat it to about 600° C. in order to break up silicic acid, iron sulphates, and similar undesirable impurities. The leaching of this product with water would give a good zinc sulphate solution, but after electrolytic precipitation the dilute sulphuric acid solution, containing some zinc sulphate, would not be sufficiently concentrated to treat the ore for the first

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DEPARTMENT OF THE INTERIOR
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BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

EXTINGUISHING AND PREVENTING OIL AND GAS FIRES

BY

C. P. BOWIE

WASHINGTON
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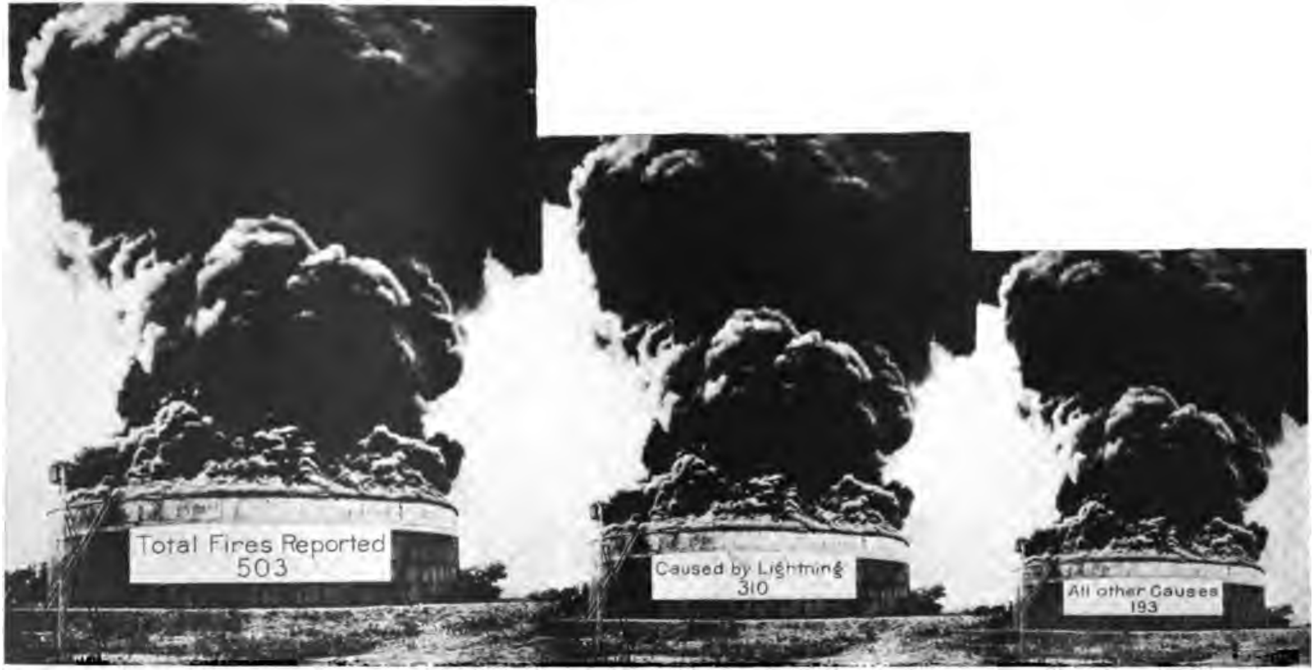
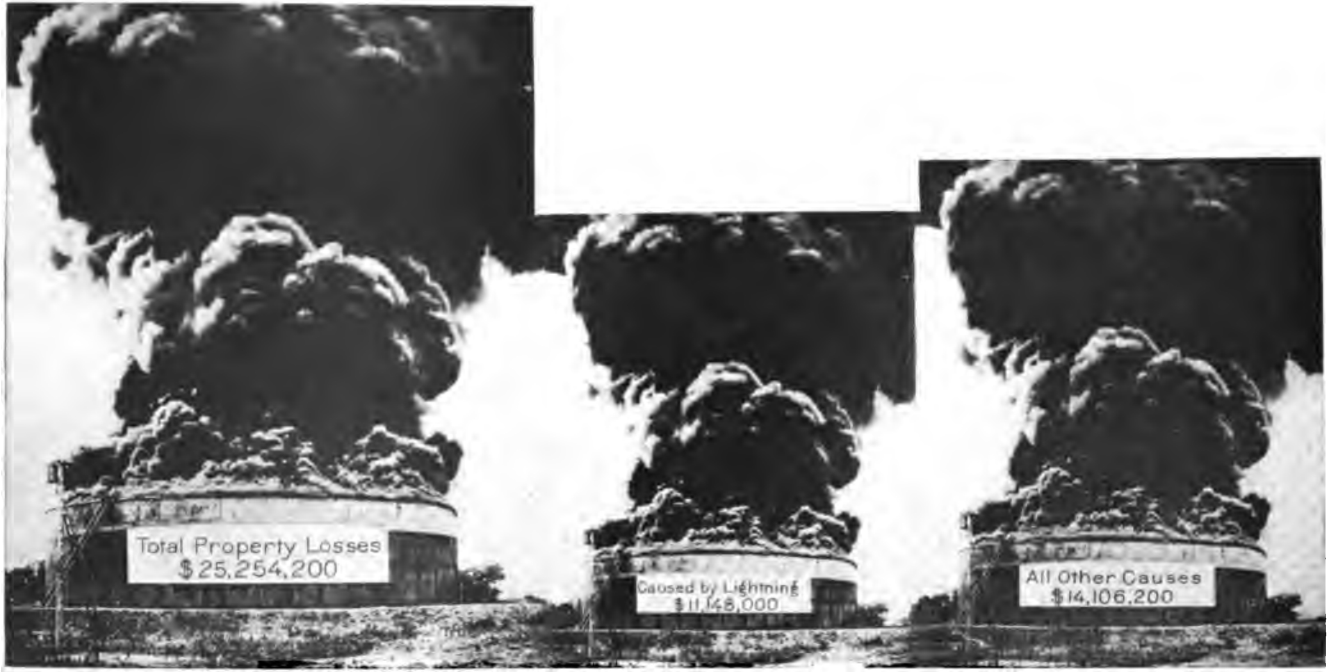


NUMBER OF OIL AND GAS FIRES REPORTED IN THE UNITED STATES FOR 10 YEARS, 1908–1918, SHOWING NUMBER CAUSED BY LIGHTNING AND NUMBER FROM ALL OTHER CAUSES.



PROPERTY LOSSES IN THE UNITED STATES FROM OIL AND GAS FIRES, 1908–1917.

EXTINGUISHING AND PREVENTING OIL AND GAS FIRES.

By C. P. Bowie.

INTRODUCTION.

During the period of 10 years from January 1, 1908, to January 1, 1918, approximately 12,850,000 barrels of oil and 5,024,506,000 cubic feet of gas were destroyed by fire in the United States, entailing a total estimated property loss of $25,254,200. During this period 503 fires[a] were reported. As is shown in Plates I and II, 310 of these fires were caused by lightning and 193 by all other causes. The losses from the fires caused by lightning were estimated to be $11,148,000, and from those due to all other causes $14,106,200. Directly and indirectly, the fires resulted in the deaths of nearly 150 persons and were responsible for almost as many more being permanently disabled.

For the past three years the Bureau of Mines has been conducting investigations to determine the nature and the specific causes of these fires, with a view to suggesting means whereby they may be successfully combatted and, in some places, perhaps eliminated entirely.

As the subject of oil and gas fires is necessarily broad, no attempt is made in this bulletin to treat it exhaustively; rather the bulletin aims to point out what has been done by operators in the past, and to describe various fire-prevention methods and fire-fighting apparatus which are being used or adopted by many of the larger oil companies. These methods and apparatus, it is believed, if universally employed by operators, will largely decrease the present enormous annual losses.

In the discussion of oil-well fires, the methods used in extinguishing several of the larger and more perplexing ones are described. As the right procedure for extinguishing any oil-well fire must be determined largely by local conditions, it is not probable that all of the methods used at some one fire, as described, will be applicable to a future fire. However, the operator, from these descriptions, may at least gain suggestions that will greatly assist him in fighting similar fires.

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[a] Not including small gasoline fires.

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CAUSES OF OIL FIRES.

Table 1 shows the losses from the more noteworthy fires at oil and gas wells during the 10-year period 1908 to 1917 inclusive. It is interesting to note that only 3 fires were reported in the Mid-Continent and the Gulf fields during January and only 6 in December for 10 years, whereas 75 were reported for June; and that during June 73 out of 92, or nearly 80 per cent, of the total number of fires reported from all fields were caused by lightning.

TABLE 1.—*Noteworthy oil and gas fires*[a] *in the United States during the 10 years, 1908 to 1917, inclusive.*[b]

	Number of fires reported.	Location.		Causes.		Estimated loss.		Estimated property losses.		Total.
		Mid-Continent and Gulf fields.	Other districts.	Lightning.	Other causes.	Oil, barrels.	Gas, cubic feet.	By lightning.	By other causes.	
January	8	3	5	....	8	62,000	950,000,000	....	362,500	362,500
February	23	15	8	10	13	233,000	390,500,000	114,700	865,200	979,900
March	33	26	7	14	19	357,200	....	124,800	443,550	568,350
April	32	25	7	22	10	610,350	....	514,200	368,000	882,200
May	46	39	7	31	15	1,614,000	1,041,000,000	1,061,700	1,372,000	2,433,700
June	92	75	17	73	19	2,334,500	800,500,000	2,741,220	876,000	3,617,220
July	72	49	23	45	27	2,580,600	147,006,000	2,138,330	3,676,450	5,814,780
August	80	71	9	69	11	3,262,400	550,000,000	3,096,000	1,030,500	4,126,500
September	44	35	9	32	12	693,250	....	781,050	1,374,000	2,155,050
October	31	17	14	12	19	535,400	....	241,000	1,190,000	1,431,000
November	20	13	7	2	18	510,600	1,125,500,000	335,000	1,466,000	1,801,000
December	22	6	16	....	22	56,200	20,000,000	....	1,082,000	1,082,000
Total	503	374	129	310	193	12,849,500	5,024,506,000	11,148,000	14,106,200	25,254,200

[a] Not including small gasoline fires.

[b] Data compiled from technical journals, personal inquiries, and other sources.

STEAM SMOTHERING LINES.

Hence the common practice is to keep water away from burning oil tanks and to fight the fire with steam fed into the top of the tank through small pipe lines 2 or 3 inches in diameter. The function of the steam is to fill that part of the tank above the oil and thus exclude air from it. Although this system has been successfully used in extinguishing numerous tank fires the writer believes that it should not be relied on as a sufficient means for overcoming such fires. If a tank has a wooden roof the steam may repeatedly extinguish the fire, and the latter may be as often reignited by embers falling from the roof; even if a tank has a steel roof, on steel roof supports, an explosion of the gases above the oil in the tank may rupture the roof, as often happens, and the steam may escape through this hole and prove wholly ineffective.

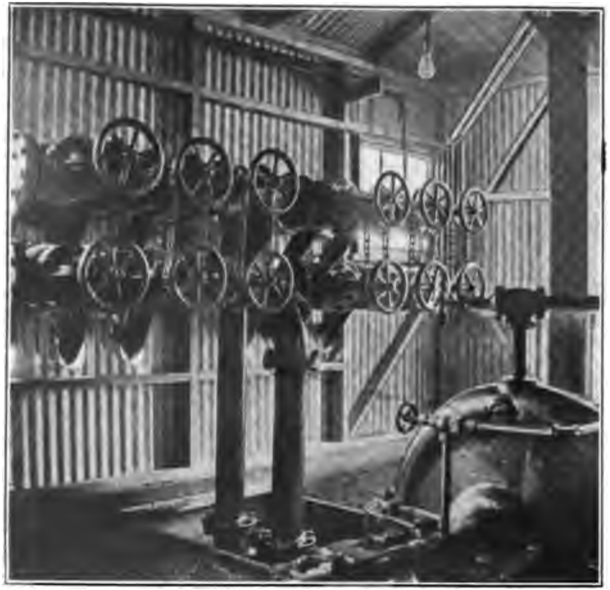
FROTHY-MIXTURE SYSTEM.

To overcome this defect in the application of steam a so-called frothy-mixture system of fire fighting has been devised, and is now being installed by many of the larger oil companies. As the writer believes that this system is to play an important part in reducing the number of future oil fires, a complete installation put in by the Associated Pipe Line Co., at Coalinga, Cal., is described here in detail, and the most important parts of the system are discussed in an endeavor to point out weaknesses and suggest improvements for future installations.

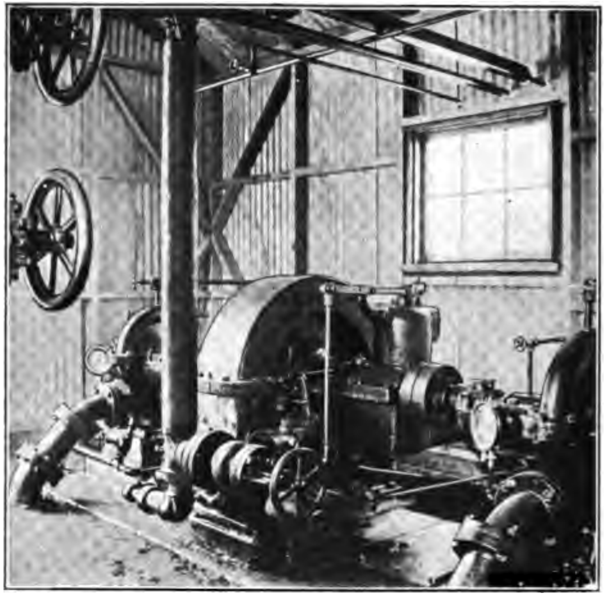
Essentially this system of fighting oil fires provides for bringing together two chemical solutions and for spreading over the surface of burning oil the thick tenacious foam resulting from their combination, thereby excluding air and extinguishing the flame. The apparatus required in practice comprises in brief two tanks—one for each chemical solution—a suitable pump, pipe lines to carry the solutions to the oil tanks or reservoirs, mixing chambers where the chemicals can combine, and a means for properly distributing the foam.

FOAMING SOLUTIONS.

In general, the solutions for producing the foam may be any two compounds that on coming together form an abundance of relatively tough bubbles inflated with a noninflammable gas. The chemicals must be fairly cheap; also they must not deposit any appreciable amount of sediment after having been in solution for a considerable period of time. Usually the chemical constituents of each solution are so proportioned that when the two are combined in equal parts a maximum amount of foam is produced. The



A. FROTHY-MIXTURE PUMPING UNIT USED AT COALINGA, CAL.



B. FROTHY-MIXTURE 5-INCH CROSS-VALVE MANIFOLD USED AT COALINGA, CAL.

proportion of froth to solution should be about 6 or 8 to 1; that is, combining 50 c. c. of each solution should produce 600 to 800 c. c. of foam. In the installation at Coalinga the following formula was used:

Solutions used at Coalinga.

SOLUTION A.	Parts by weight.
Water	100
Aluminum sulphate (crystal)	10
Sulphuric acid, 66° B	½

SOLUTION B.	Parts by weight.
Water	100
Ground glue	1¼
Glucose	½
Sodium bicarbonate	7½
Arsenious oxide	$\frac{1}{32}$

CAPACITY OF SOLUTION TANKS.

The installation was designed to protect four 55,000-barrel oil tanks each 114 feet 6 inches in diameter, one 37,000-barrel tank 95 feet 6 inches in diameter, one 30,000-barrel tank 86 feet in diameter, and two fuel-oil tanks each about 8 feet in diameter. On an estimate of 6 gallons of foam for 1 gallon of solution, to produce over the surface of all the various tanks a blanket of foam 5 inches thick in 5 minutes required the delivery of each solution at the following rates:

Solution required for each tank.

	Gallons a minute.
55,000-barrel tank	550
37,000-barrel tank	368
30,000-barrel tank	308
Fuel-oil tanks, about	6

On this basis 6,200 gallons of each solution would be needed to cover all the tanks once with 5 inches of foam. In order to provide an ample supply of solution and to allow a reasonable factor of safety, it was determined to use solution tanks 15 feet in diameter by 11 feet 9 inches in height, each having a capacity of 15,700 gallons.

The tanks were made from No. 10 U. S. gage galvanized iron, all seams riveted and soldered. A 6-inch suction outlet was placed on the side of each tank 3 inches from the bottom. The tank for solution B—the soda solution—had also a 3-inch drain on the bottom. The tank for solution A—the acid solution—was lined with sheet lead weighing 8 pounds per foot, and in lieu of a drain had in the bottom, immediately beneath the hatch, a depression made by riveting on a pan ½ inch deep and 12 inches in diameter. When the tank has to be emptied and cleaned, the liquid below the 6-inch side outlet can be removed with a steam ejector, or siphon, having its suction end placed in the pan. This method was considered cheaper and more satisfactory than using draw-off pipe, as such pipe would have to be of lead or lead lined.

The tops of the tanks were made of wood covered with roofing paper, the rafters being supported by the tank shell. The tanks were set on a platform made of 4-inch by 12-inch redwood planking laid on mudsills resting on an earthen grade built up about 3 feet above the pump foundation so that the solutions might flow to the pumps by gravity, even when the tanks were almost empty.

PUMPS.

For delivering the solutions to the various oil tanks two double-suction, single-stage, horizontal-shaft, centrifugal pumps are used directly connected to a 110-horsepower, impulse-type, steam turbine, the pumps and turbine being mounted on one bedplate. The pumping unit, shown in Plate III, was designed to deliver 550 gallons a minute against a head of 200 feet with a steam pressure of 140 pounds per square inch. The working parts of the soda-solution pump were of cast iron; the impeller and wearing rings of the acid-solution pump were of "acid bronze."

PIPING SYSTEM.

A separate system of piping controlled at the pump house, shown in Plate IV, carries each solution to its tank, each pump being so connected as to discharge into a manifold (Pl. III, *B*) composed of five 5-inch cross valves placed end to end. The soda-solution manifold is placed immediately above the acid-solution manifold, the corresponding valves to each system of piping being opposite each other. The stems of these valves are so connected with sprocket chains that the operator in opening the valve that permits the acid solution to flow to any given storage tank is opening, at the same time and at the same rate, a valve that admits soda solution to the same tank. This arrangement of sprocket wheels and chains is shown in Plate III, *A*. By this means the delivery of a proper proportion of each solution is insured whenever pumping starts. Each valve is plainly marked with the number of the oil tank to which it supplies solution, so that the operator has complete control of the entire piping system and can watch the pumps while remote from the fire.

The pipe used was ordinary merchantable galvanized-iron pipe 1 to 4 inches in diameter, depending on the required amount of liquid to be carried. The 1-inch pipe, which is taken off the ends of the manifold, serves the fuel tanks. Because of the relatively high proportion of alkaline salts in the soils at this station, all the pipes that

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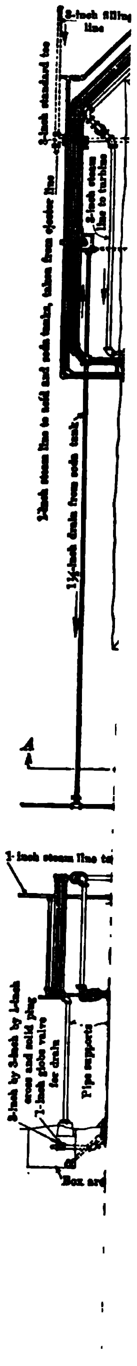

3-inch filling line
3-inch standard tee
3-inch steam line to acid and soda tanks, taken from ejector line
3-inch steam line to turbine
1½-inch drain from soda tank
A
1-inch steam line to
3-inch by 3-inch by 1-inch cross and solid plug
1-inch globe valve for drain
Pipe supports
Box ar

were buried were treated with a coat of refined asphaltum, wrapped with asbestos roofing paper, and then again coated with asphaltum before they were covered with earth. All lines are carefully laid on true gradients. At the low points are drains, so that after any part of the system has been used the solution remaining in any pipe can be drawn off and returned to its tank; the lines can then be flushed and the water drained off, leaving them empty when not in use.

The 6-inch suction nozzle from the acid tank is lead lined to the suction valve, about 8 inches from the tank. The valve itself, shown in Plate V, was specially constructed from a stock Crane 6-inch flanged cross valve by replacing the disk, seat, collar, and valve sleeve with fittings made of an acid-resisting metal that is about 5 parts antimony and 95 parts lead; hence when the valve is closed, as it is except when the system is in actual operation, all the parts that come in contact with the acid solution are acid-resisting. To guard against acid solution entering the iron piping if the valve leaks a $\frac{1}{2}$-inch drain is provided in the bottom, which is kept open when the system is not in use. The water suction for flushing out the system is 3 inches in diameter and is connected to one run of the valve on the soda tank, as shown in Plate IV.

FOAM MIXING BOXES.

For each oil-storage tank there is a rectangular mixing box made of $\frac{3}{16}$-inch steel plate into which the pipe lines leading to that tank discharge. Each box on the outside of the tank is supported by the shell and extends about 1 foot above the top angle iron, as shown in detail in Plate VI. Plate VII, *A* shows a mixing box and the solution lines leading into it. The object of having the boxes on the outside of the tanks is that if the tank roof should be wrecked by an explosion of gas at the outset of the fire, the boxes and piping would be unharmed and would still be able to deliver foam upon the blazing surface of the oil. The storage tanks have wooden roofs and the foam is admitted through a raised hatchway (see Pl. VII, *B*) built up about 2 feet above the original roof.

The boxes for the 55,000-barrel tanks are 4 feet 5 inches long by 2 feet 9 inches wide by 2 feet 10 inches deep, inside measurements. The upper 1 foot of the side toward the tank is cut away, and a lip $6\frac{1}{4}$ inches wide is riveted to the upper edge of this short side. This lip rests on top of the top angle iron of the tank. The notch, with the lip and a sheet-iron apron about 18 inches long, shown in Plate VI, form a wier and a flume through which the foam is delivered into the tank.

The boxes for the 37,000 and 30,000-barrel tanks are of similar construction, but of smaller dimensions. Those for the fuel-oil tanks are merely galvanized-iron pans about 14 inches square by 10 inches deep. The box is placed near the center of the tank and immediately beneath the roof, being supported by the solution pipes themselves.

The solutions enter the rectangular boxes on one side near the bottom and discharge into headers made of 6-inch pipe that has slotted perforations on the under side to mix the solutions thoroughly before being discharged over the wier into the tank.

FOAM SLIDES.

To insure the foam reaching the surface of the blazing oil at whatever depth it may stand in the tank, without the bubbles being destroyed by the impact of the drop, a series of baffles were designed. Their construction is shown in detail in Plate VI. The baffles are made of No. 10 U. S. gage black iron and are supported on 2 by 2 by ¼ inch angle irons which rests on a shoe at the bottom of the tank. These angle irons were each made in two lengths in order to facilitate their being placed in position, as the system was installed years after the tanks had been built. Each leg is fastened to the shell of the tank at a point about 10 feet from the bottom by a 2 by 2 by ⅜ inch angle.

The foam pouring from the mixing box falls upon the uppermost baffle plate. The sticky character of the foam retards its descent, and it reaches the bottom of the tank, after having passed over the series of baffles at a velocity not much greater than when leaving the mixing box.

PREPARING THE SOLUTIONS.

In preparing the chemical compounds, each solution tank is filled with water to within about 2½ feet of the top. In the acid solution tank the water is heated with a steam jet to a temperature of 180° F., and the aluminum sulphate and sulphuric acid added. While the chemicals are being added, the contents of the tank are thoroughly agitated with compressed air. In preparing the soda solution, the ground glue is first added slowly to the water before heat is applied, the liquid being constantly agitated with air; then the temperature is raised to 160° F., the glucose and the bicarbonate of soda are put in, and the mixture is agitated until all three are completely dissolved. The arsenious oxide is added last. Both tanks are then filled to the top with water and agitation continued until no undissolved chemical can be detected in the bottom of the tanks by stirring with a long pole.

TESTING THE SOLUTIONS.

The reagents, particularly the glue in the soda solution, slowly decompose so that it is necessary to add more from time to time.

For this reason the solutions are tested about once a month by mixing together 50 c. c. of each in a graduated beaker. If the volume of foam produced is less than 600 c. c., the necessary components to bring the solutions up to specification are added.

TRYING OUT THE SYSTEM.

No fires have occurred in the plant since the system was installed. However, it has been thoroughly tested out on several occasions. Foam can be delivered into the farthest 55,000-barrel tank, 470 feet from the pump house, in 1⅔ minutes after the pumps are started, and four minutes later the entire surface of the oil will be covered with foam about 5 inches in depth. The foam is composed of minute bubbles and is so tough and durable that a considerable quantity will be found on the surface of the oil 24 hours after it has been spread.

CARE OF SYSTEM.

As it is absolutely essential that such a system be kept in first-class condition, thorough inspection is made once a week. Each part is examined and tested to make sure that it is ready for successful operation at a moment's notice. Steam is turned into the turbine, and the pumps are run for a few seconds, all valves are "cracked" and again closed to insure against the possibility of any valve being stuck at the crucial moment, and the mixing boxes are examined to see that they are clear of all objects that might impede the flow of foam.

COST.

The system was installed during 1914. Its cost complete, including turbine, pumps, piping, tanks, labor, etc., was approximately $8,500, or a little less than 3 cents per barrel capacity of all the tanks protected. As this plant is an oil-pumping station, steam is always kept up, and it is not necessary to employ a man solely to operate the foam pumps. The cost of maintenance, therefore, is practically nothing, and the depreciation will be very small for a number of years to come.

DISCUSSION OF "FROTHY-MIXTURE" SYSTEMS.

The installation at Coalinga has been described in detail because it has many commendable features, especially where only a few tanks, not very far apart, are to be protected. The pump house is only a few feet from the boilers. The valves are all within a radius of a few feet, and by opening four valves (two at one operation) the solutions are started on their way to whatever tank desired. This system, if need be, can be operated by one man, and eliminates the need of opening field valves, where there is always danger of a mistake, particularly at night.

FOAMING SOLUTIONS.

Beside the formula used at the Coalinga plant, given on page 11, the following have also been used:

Formulas for foaming solutions.

FORMULA 1.

SOLUTION A.	Parts by weight.	SOLUTION B.	Parts by weight.
Water	100	Water	100
Aluminum sulphate (crystal)	14	Sodium bicarbonate	9¼
Extract of licorice (powdered)	3		

FORMULA 2.

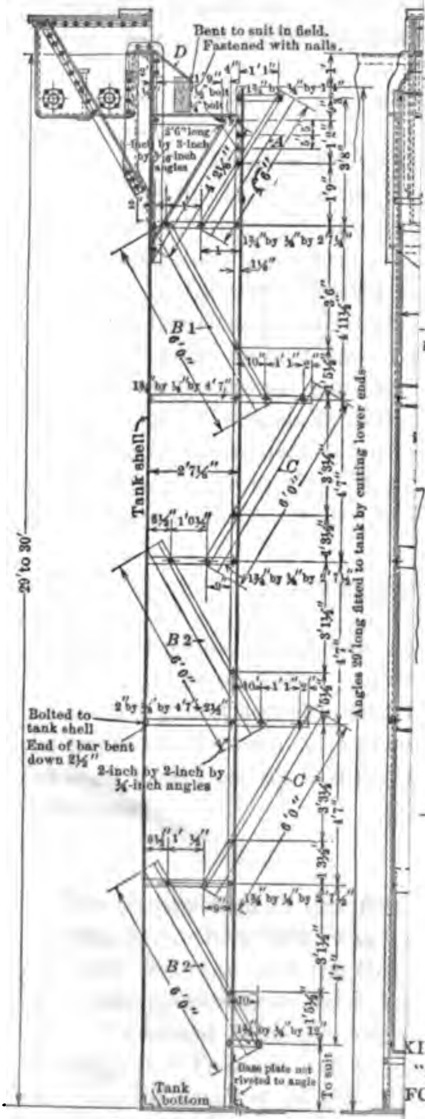
Water	100	Water	100
Aluminum sulphate (crystal)	12	Sodium bicarbonate	10
Acetic acid	½	Ground glue	1
Ground glue	1	Glucose	¼
Glucose	¼		

Another formula that is being extensively used has for one of the components of the soda solution (solution B) a so-called secondary extract of licorice root. The licorice takes the place of the glue, and, so it is claimed, will remain in solution indefinitely and will not deteriorate with age.

SOLUTION TANKS.

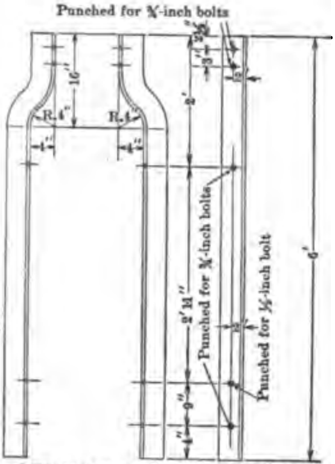
The capacities of the solution tanks depend on the number of square feet of oil surface to be covered. On an estimation that 1 gallon of mixture (one-half gallon of each solution) will produce 7 gallons, or 1,617 cubic inches of foam, to cover the oil in a 55,000-barrel tank with 6 inches of foam would require about 2,800 gallons of each solution. To this figure, of course, a reasonable safety factor should be added. Where the number of oil tanks to be protected is small, as in the installation described, the size of the solution tanks can be figured with a reasonable degree of accuracy. However, where hundreds of oil tanks are involved, as in a large storage farm, the sizes of the solution tanks become a matter of judgment, for it is unreasonable to suppose that all of the oil tanks, or even a majority of them, would become ignited at the same time.

The sulphurous solution is a weak acid, and consequently will react on iron or steel. Hence the acid-solution tank must be made of wood or lined with some acid-resisting material, such as enamel or lead. In the judgment of the writer, lead is superior to enamel, be- use it is flexible, and settling of the tank or blows on the shell will

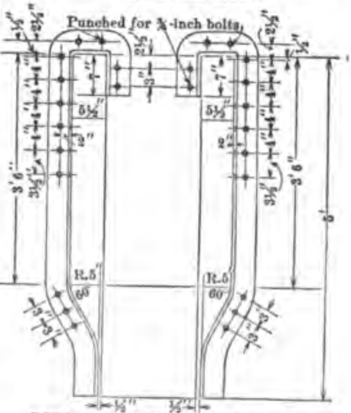


SIDE ELEVATION OF BAFFLES

FR



DETAIL OF 3-INCH BY 3-INCH BY 5/16-INCH ANGLE-IRON HANGERS FOR FOAM SLIDE



DETAIL OF 4-INCH BY 4-INCH BY ⅜-INCH ANGLE-IRON HANGERS FOR MIXING BOX

KING BOX AND FOAM SLIDE
"FROTHY MIXTURE" SYSTEM
FOR 55,000 BARREL TANKS
Pl. V.

not crack a lead lining. However, lead is usually more expensive as to first cost.

LIME WATER TANK.

Some installations include a tank of about 2,000 gallons capacity, containing lime water for flushing the acid solution lines after use.

MIXING TANK.

Where the solution tanks are large, a fourth tank of about the same capacity as the lime tank is often installed for mixing the chemicals, as thorough mixing can be accomplished more easily in a small tank than in a large one. After one solution has been mixed, the tank is thorough cleaned and the other solution is mixed.

It has been suggested that if two such tanks are provided, each having proper connections to the solution tanks, there is no reason why fresh solutions can not be mixed, if need be, during the progress of an unusually large fire. This suggestion seems a pertinent one, as it solves the problem of providing an ample supply of chemicals for protecting large tank farms without the necessity of tying up an undue amount of capital in solutions and in containers for them.

PROTECTION FOR SOLUTION TANKS.

The solutions will freeze at a temperature somewhat below the freezing point of water, probably at about 25° F. In cold climates, therefore, protection against freezing is necessary. The tanks should be housed and steam coils placed in the houses, but not in the tanks, as high local heating is injurious to the chemical components of the solutions.

PUMPS.

The pumps used in the installation described seemed to give satisfaction when the system was tried out, but they are not, in the writer's opinion, the best type for this kind of work. As the chemical solutions are proportioned with a view of using equal quantities of each, it is important that the pump accurately meter the liquid passing through it. This can not be done with centrifugal pumps. A slight increase of friction in the discharge line from one pump over the discharge from the other will be taken up by slippage in the pump, and unequal quantities of the solution will be delivered at the mixing box. This excess of one of the solutions is entirely wasted.

A duplex type of crank and flywheel pump, or a duplex power pump, using one cylinder for the acid solution and the other for soda, are much better adapted to this service, as they will measure the quantities pumped quite accurately. Direct-acting, duplex, steam pumps are being used in some places; these are subject to the same defects, by reason of the short stroke, as the centrifugal pumps, although in less degree.

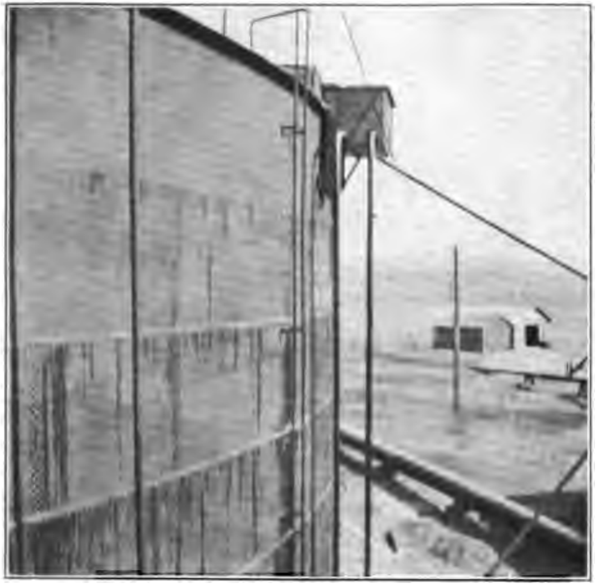
PIPE LINES.

The system of having separate piping for each tank, controlled at one central station, is in many ways an ideal one for installations of this character, but the cost will become prohibitive when a large number of tanks are to be protected. Where such is the condition, main headers of 4 to 6-inch pipe are run from the pump house to various parts of the tank farm, or refinery, and from these, at points opposite the respective tanks, smaller lines convey the solutions to the various mixing boxes, as illustrated in Plate VIII. All valves are kept closed when the system is not in use. When a fire starts, it is necessary to open field valves in the vicinity of the blazing tank before foam can be turned into it. However, these valves are, or should be, at a safe distance from all tanks, so that there can be no danger to an employee from opening them when the tank they supply is on fire. Also, any large tank farm or refinery should always have on the premises a number of trained fire fighters, who are familiar with the valves and the pipe lines. It is the practice of most companies to have the valves and the pipe lines—wherever they are above ground—carrying the solutions painted in different colors, as bright yellow and bright green, so that wherever a light can be thrown upon them they are at all times unmistakable. All tanks should be plainly numbered in large figures, with the corresponding numbers on the valves admitting the solutions to them. This is especially important where the valves are placed in clusters.

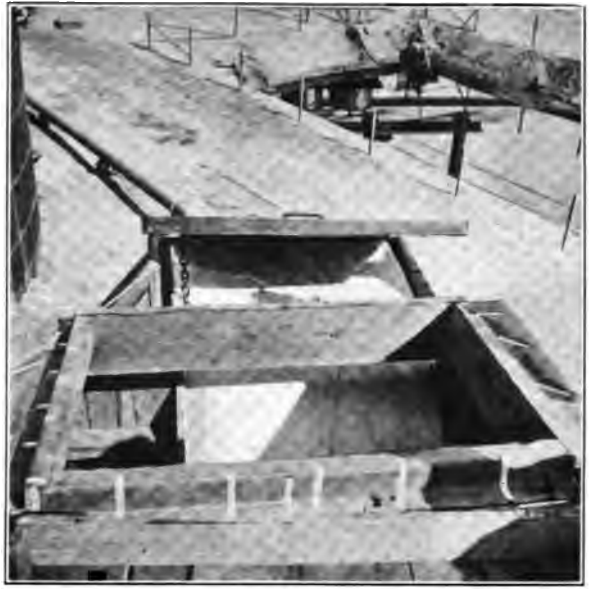
MIXING CHAMBERS.

The mixing chambers used are of various types. Plate IX, *A*, shows a type in use by a Texas company. This chamber consists of a cast-iron return bend with one leg longer than the other. The bend is hung over the top of the tank shell, and the solutions enter by separate pipes at the bottom of the longer leg, where they are mixed before being discharged into the tank.

Another type, shown in Plate IX, *B*, is made from light sheet iron, and is hung to the shell of the tank on the outside. The foam is admitted into the tank just below the top angle iron through a lip or nozzle about 8 inches long. The solutions enter the bottom of the chamber as in the type previously described. The tank shown was under construction and the foam pipes had not been installed.



A. MIXING BOX OF FROTHY-MIXTURE SYSTEM AT COALINGA, CAL.



B. FOAM FLOWING INTO TANK THROUGH HATCHWAY, FROTHY-MIXTURE SYSTEM, COALINGA, CAL.

Still another type is a standpipe about 12 inches in diameter placed inside the tank near the center and extending up to a point a little above the top angle iron. The solutions are admitted into the bottom of this pipe through the small pipes and are allowed to mix as they pass upward.

The essential points about the mixing chambers are: They should be so placed about the tank as to be reasonably immune from injury should the roof of the tank be wrecked by an explosion; they should be large enough to permit thorough mixing of the solutions before discharge; and they should be so constructed that at whatever height the oil stands in the tank there is no possibility of the discharge into the tank being so shut off that the foam can not enter freely. As to size, the boxes used on the 55,000-barrel tank in the installation described seem by actual trial to be amply large. When the different solutions are brought together the reaction is practically instantaneous, hence the boxes are probably larger than is necessary. However, this is an error on the side of safety; a slight excess in capacity can in no way be detrimental, and insures thorough mixing of the solutions.

FOAM SLIDES.

Foam slides have been omitted from all of the more recent installations, as experience has demonstrated that the bubbles are elastic enough to remain intact even though they are subjected to a sheer drop of 20 feet or more.

AGITATING THE SOLUTIONS.

In order to insure uniform and thorough mixing of the solutions at all times, each tank should be equipped with a means for agitating the liquids. This can best be done by running an air line down into the tank at the center, terminating in a side outlet cross resting on the bottom of the tank. In each horizontal arm should be placed a short piece of pipe. Air should be turned into this pipe about once a week, and the contents of the tank thoroughly agitated.

FROTHY-MIXTURE SYSTEMS IN GENERAL.

Aside from the manually operated systems several types of automatic systems are in use. However, as these all work on the same general principle as the one described, so far as the matter of extinguishing the fire is concerned, they will not be described at length. They depend on a fusible link, or plug, to start the flow of the foam, which is maintained either by gravity or by gas pressure. Automatic systems would probably prove highly satisfactory for small installations, where only two or three isolated tanks are to be protected, but the cost of construction, as compared with manual control, becomes excessive where a large number of tanks are involved.

AN OIL FIRE EXTINGUISHED BY THE FROTHY-MIXTURE METHOD.

The following is an abstract from a report by Supt. J. W. Tryon, concerning a fire which occurred at the Gulf Refining Co.'s plant at Port Arthur, Tex., in the summer of 1916:

The installation at our Port Arthur refinery consists of four steam-operated foam-pumping stations, each equipped with duplex pumps built especially for the service, handling 367 gallons per minute of each solution and two tanks—one for each solution—of about 600 barrels capacity. The pump stations are so located that each one serves about the same area. After leaving the pumps, the solutions pass through 6 and 4 inch main header lines, which are tapped by 2-inch lines opposite the various tanks that convey the chemicals to the respective foam boxes.

The pumping plants are inspected daily by our fire marshal, and are at all times maintained in perfect working order ready for immediate operation. Immediately upon the approach of a storm, certain employees are detailed to go to the pump houses and start the pumps, which are equipped with by-passes relieving back into the tanks through pressure valves set at 85 or 90 pound pressure. This assures prompt operation, the only thing necessary being the opening of the valves on the main line, and on the branch lines leading to the particular tank where the foam is required.

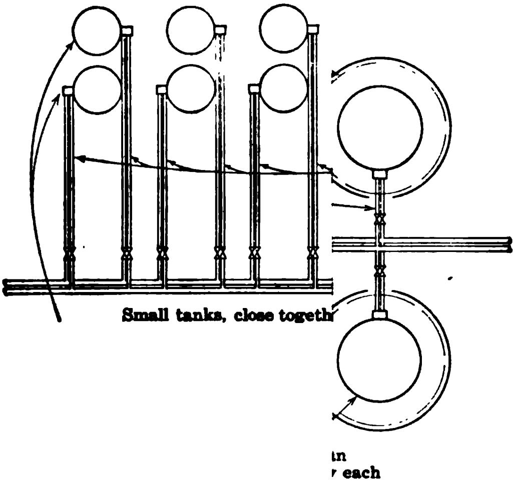
On the morning of July 19, 1916, there were indications of a storm, and the pump houses were manned as described. At 10.50 a. m. tanks Nos. 850, 855, and 857 were struck by lightning. No. 850 is 70 feet by 10 feet, having a capacity of 6,780 barrels. At the time it was struck it contained about 5,216 barrels of gasoline distillate. Tank 855 is 65 feet by 10 feet, with a capacity of 5,865 barrels, and contained 4,061 barrels of gasoline distillate. Tank 857 is 90 feet by 30 feet, having a capacity of 33,750 barrels, and contained 28,409 barrels of gasoline.

It is impossible to say whether all three of the tanks were ignited by the flash of lightning or whether No. 855 was struck and the fire communicated by it to the other tanks. An explosion occurred in No. 855, which blew the roof about eight feet into the air. In falling it broke in the center, and most of it settled beneath the contents of the tank; however, some of the wooden rafters extended over the edge.

The instant after the explosion the entire contents of tank No. 855 were blazing furiously. The roof of tank No. 850 appeared to be intact, but was burning all around the eaves. The roof of No. 857 was cocked up in several places and was burning fiercely.

Immediately after the tanks were struck the pumper opened the valves on the main lines, the fire marshal opened the valves to No. 857 and the foam was started into this tank. At 11 o'clock, 10 minutes after the fire started, there was no evidence of fire in tank No. 857. The foam was then started into 850 and 855, and the fires in both promptly extinguished—855 was entirely under control by 11.15 and 850 by 11.20.

The steps leading to tank No. 855 were not burned away, and an excellent opportunity was afforded to watch the action of the foam. Everything worked exactly as we had expected it to work. An excellent heavy foam poured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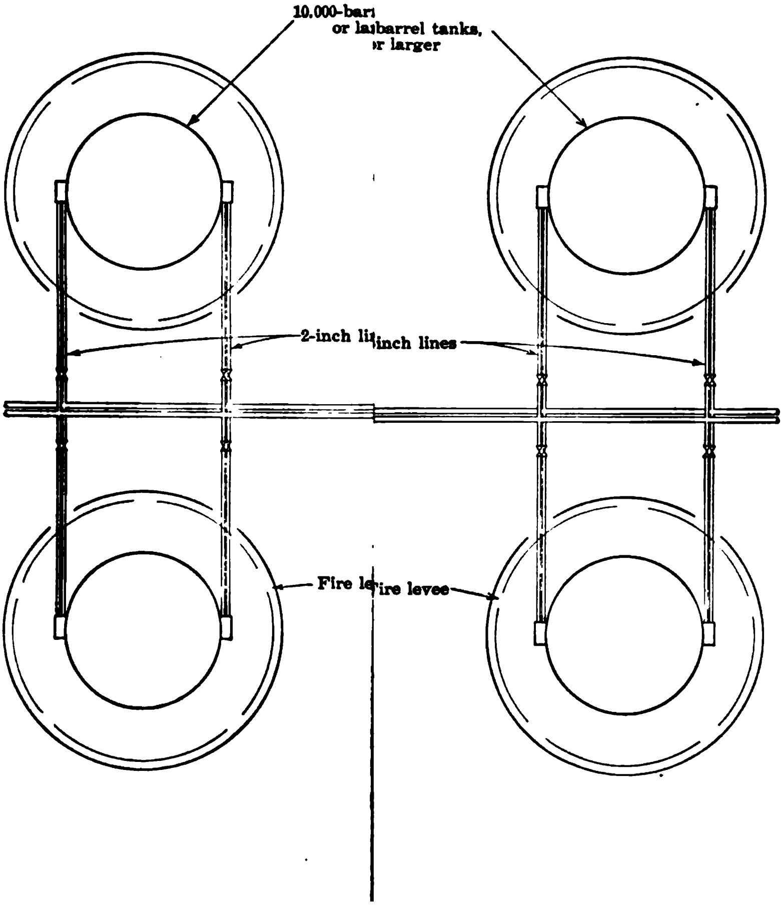
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out from the mixing boxes in great volume, and within five minutes after starting it into tank 855 there was a blanket about 6 inches deep of it over the entire surface of the oil. This being the tank in which the explosion occurred, the flow of foam was somewhat impeded by the protruding rafters, but it gradually closed in and shut off the fire entirely. The fire in the rafters was put out by spraying water on them.

Plate X, *A* shows the condition of the foam on the surface of the oil after the fire had been extinguished and Plate X, *B* shows the outside of the tank and the wooden stairway, which was still intact. The report continues:

> This was our first occasion for using foam on an actual fire. We desired to avoid the possibility of failure, and therefore proceeded very deliberately, and, no doubt, used considerably more foam than was necessary. I believe that if it should again become necessary to use this system we will be able to reduce the time required to extinguish a tank fire and that we will also be able to reduce very materially the quantity of solutions used.
>
> Damage done by the fire was limited to two wooden roofs and some slight damage to the top rings of two of the tanks. Of the total contents of the three tanks, 37,686 barrels, there was a loss of but 191 barrels. Without the foam the minimum loss would probably have been 19,000 barrels, with a possibility of a very much larger loss. We believe this system to be an excellent practical solution of the problem of extinguishing tank fires if it is properly installed and operated.

ALL-STEEL TANKS.

The reader will notice that the tanks that Mr. Tryon has described had wooden roofs covered with roofing paper. So far as the writer is able to determine, all of the tanks that have been ignited by lightning in the past had wooden roofs covered with roofing paper, or sheet iron in bad condition, permitting the light hydrocarbon gases to pass freely between the sheets, so that at all times a stream of inflammable gases might be rising from all parts of the roof. Many of the oil operators believe that during a thunder storm it is not necessary that a tank should be actually struck by lightning for its contents to be set on fire. The writer has talked with numerous persons who claim that they have seen, after a flash of lightning, what they described as a "pillar of fire" extending above a tank just before the tank burst into flames.

In view of the fact that storm clouds in the Mid-West fields often sweep very low, and that several instances have been noted where ignited tanks have burned quietly without the roof or shell being disrupted, it is possible that the condition described is true. It is easy to conceive of atmospheric conditions being such that a storage tank with a leaky roof would be surrounded with a highly inflammable mixture of hydrocarbon gas and air, also that the gaseous stratum might extend a considerable distance from the tank, even communicating with bodies of similar gases from other storage tanks situated at a considerable distance. Discharges of electricity from one part of a low sweeping cloud to another would serve to ignite the gas, and, all conditions being favorable, the fire would follow the stratum of gas, igniting the tank from which the gas issued, or perhaps a number of tanks.

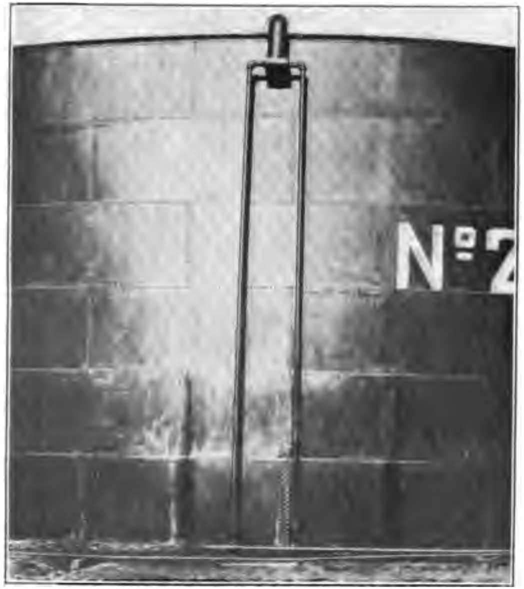
To overcome this condition, the Texas Co. at its storage farms along the Gulf of Mexico is constructing gas-tight all-steel tanks having one 12-inch vent (shown in Pl. XI, *A*) leading from the roof of the tank and terminating outside the fire levee in a standpipe about 10 feet high. On top of the standpipe is a tee having a plug screwed into one end and a short piece of pipe in the other; on the end of this pipe is a pair of flanges which hold a fine mesh wire gauze. The tank is fitted with four explosion doors, one of which is shown in detail in Plate XI, *B*. The roof of the tank being gas tight, all the gases arising from the oil are forced to pass out through the 12-inch pipe. Should the column of gases from this pipe be ignited by lightning, or by any other means, the fire would not pass through the wire gauze and could be extinguished by placing a blanket over the outlet of the pipe. If the flame should penetrate beyond the gauze into the tank, the explosion doors would relieve the pressure and on falling back into position would shut off the air and extinguish the fire. Up to the present time none of the tanks equipped in this manner have been struck by lightning. A decided improvement on this system would be to pass the gases discharged from the standpipes through a condenser, or absorber, or otherwise usefully dispose of them, thus eliminating the possibility of fire from vapors escaping into the atmosphere, besides saving the gases.

The writer believes that an all-steel tank equipped in this way, and having explosion doors and a proper "frothy-mixture" system if ignited by lightning or otherwise, could almost surely be saved with practically its entire contents. If it were not for the possibility of an explosion of gas disrupting the roof, steam would doubtless be as effective as foam.

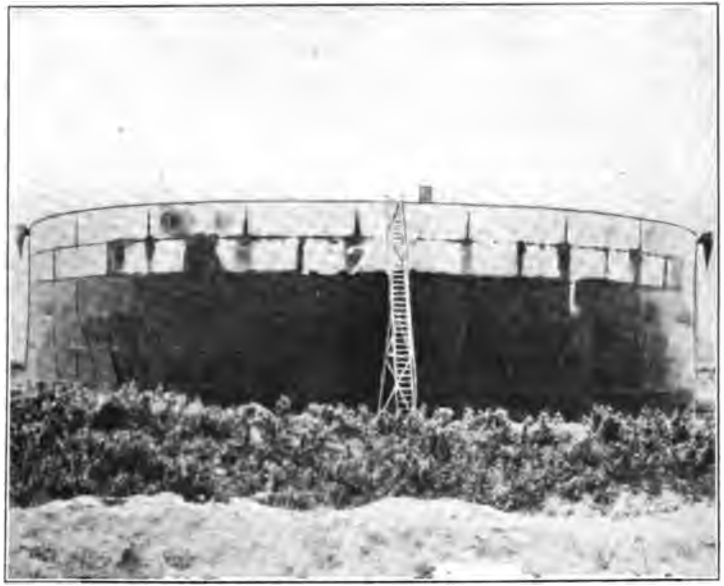
Sometimes tanks without gas-tight roofs catch fire at the leaky places, such as the eaves, and burn for several moments before the oil itself bursts into flame. Such fires have been extinguished by smothering the blaze with blankets. However, this practice leads the fire fighters into great danger and should be condemned.

EXTINGUISHING GAS-WELL FIRES.

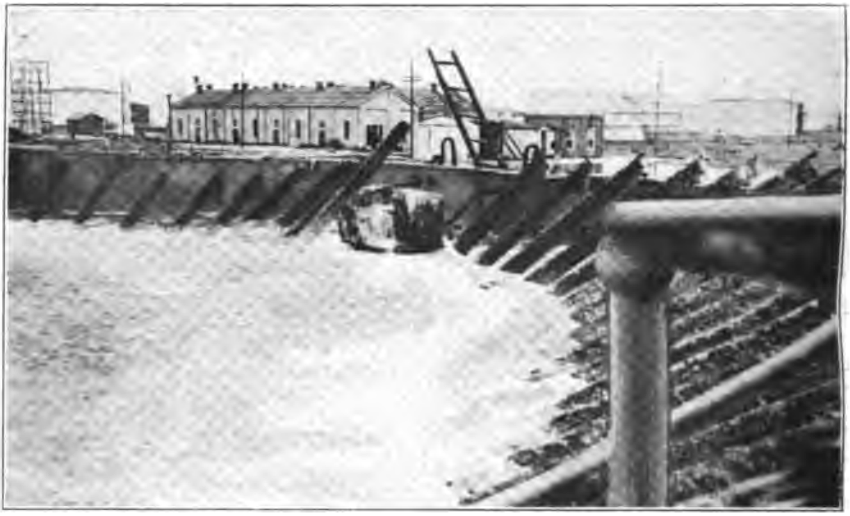
Fires at gas-wells, except at wells producing large volumes of gas, are, as a rule, not particularly difficult to overcome if the gas can be confined in one stream, because if the flow can be cut off for an instant the flame will be extinguished. Control of the gas flow, however,



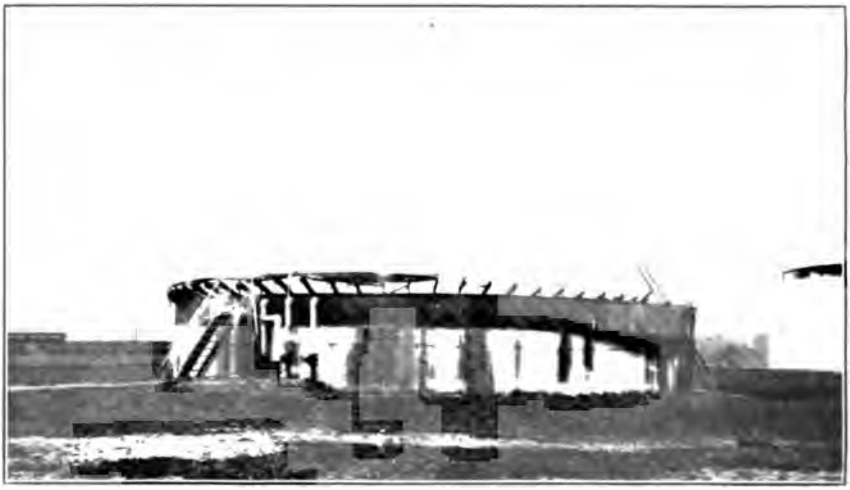
A. TANK HAVING MIXING CHAMBER MADE FROM CAST-IRON RETURN BEND.



B. TANK HAVING MIXING CHAMBER MADE OF SHEET IRON.



A. CONDITION OF FOAM ON SURFACE OF OIL AFTER FIRE HAD BEEN PUT OUT.



B. OUTSIDE OF TANK AFTER FIRE HAD BEEN PUT OUT.

Note the wooden stairway still intact.

is sometimes very difficult, as is shown[a] by a fire at a gas well of the Standard Oil Co. of California near Taft, Cal., in July, 1916. The well was making about 20,000,000 cubic feet of gas daily and was being brought under control at the time it caught fire. A gate valve had been placed on one of the strings of casing; gas was escaping through the gate valve and on the outside of this casing. The casing beneath the valve, owing to the heat of the fire and the weight of the valve, partly failed, permitting gas to escape laterally at high velocity through numerous cracks. The flame was then so large that it could not be smothered with steam. In order to bring the gas into one stream again, the casing was cut beneath the gate valve with a rifle, about 40 shots being required.

When the gas flow has been confined in one stream the flame, unless it is very small, will always be some distance above the casing. The greater the pressure on the gas the longer this column of unburned gas will be. If enough steam or water can be directed against this part of the gas column to interrupt the stream of ascending gases momentarily even the largest fire will be put out. As there are no particles of blazing fluid dropping from the flame, and as the casing head and the ground in the vicinity of the well will seldom be hot enough to reignite the gas, there is little danger of the flame being rekindled. However, it is a wise precaution to thoroughly spray the casing head and the ground adjacent to it with water just before the steam or water is turned onto the gas column.

STEAM METHOD.

In putting out gas-well fires with steam, the usual practice is to set up portable field boilers in the vicinity—sometimes as many as 20 are required—and surround the burning well with steam pipes. The pipes terminate in goosenecks having discharge ends so flattened as to throw a fan-shaped spray of steam against the gas column. The lines are usually screwed together at a safe distance from the well and pushed into position, or the men are protected from the heat by sheet-iron or asbestos shields, and water is continually sprayed on them as they work. When all is in readiness, steam is turned on from all boilers simultaneously. If the volume is sufficient, a blanket or cloud of steam will be formed and momentarily interrupt the gas stream just above the head of the casing, and the fire will be extinguished.

WATER CONVERTED INTO STEAM.

A method[b] used on a burning gas well near Monroe, La., in June, 1917, by William Guerin, formerly of the New York Fire Department, is worthy of note. The well was estimated to be making 40,000,000 cubic feet of gas daily. For the protection of the men two shields were pushed to within a few feet of the well. Two ⅞-inch jets of water under a pressure of about 40 pounds were so directed against the column of gas immediately above the casing that they converged from an angle of about 90 degrees. The nozzles were then slowly raised until the streams played on the base of the flame. Then the men handling the hose pressed their thumbs against the nozzles, making a fan-shaped spray. The spray, on striking the base of the flame, was rapidly converted into a cloud of steam which rose beneath the flame and put it out.

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[a] Editorial, Blaze of gasser out after five days' fire: Oil and Gas Jour., vol. 15, July, 1916, p. 32.

[b] Manufacturers' Record, June 28, 1917.

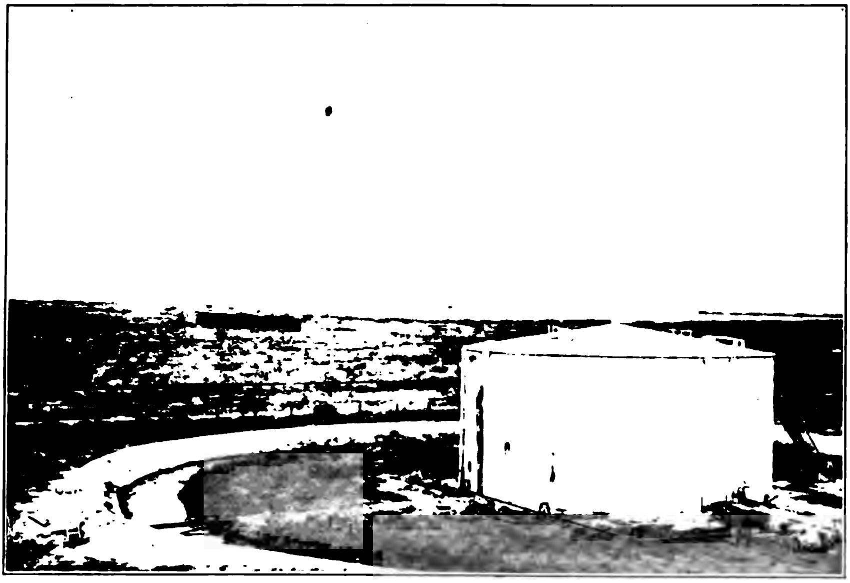
HOOD METHOD.

H. O. Ballard, of the Empire Gas & Fuel Co., has adapted the familiar snuffer principle to a portable extinguisher, which has been successfully used on small gas fires. This extinguisher is shown in Plate XII, *A*. The hood is mounted on wheels and terminates in a valve to which is attached a piece of 14-inch pipe about 20 feet long. The pipe can be extended to any desired length by screwing on more joints.

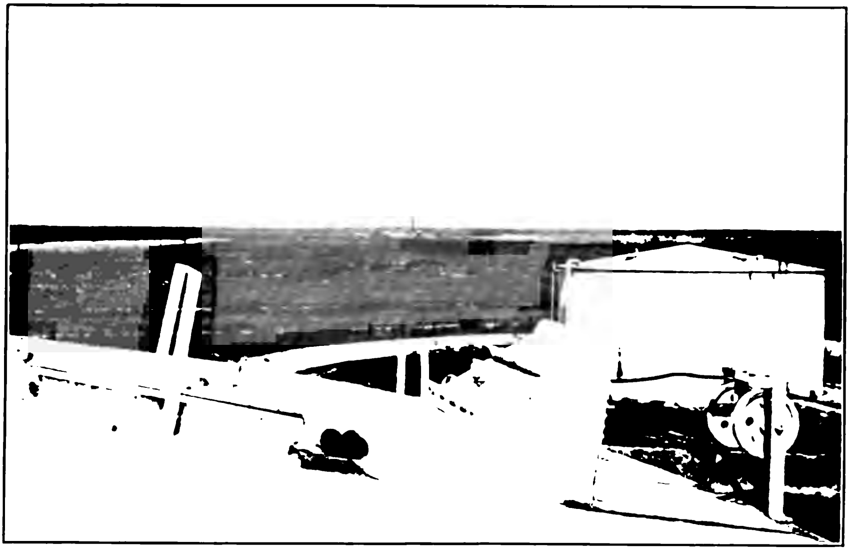
On either side of the hood, midway of its height, is a short piece of 10-inch pipe also containing a valve. The hood is taken to the site of the burning well. The desired extension is added to the 14-inch pipe, the large valve is opened, the two small valves closed. The hood is then upended over the fire by means of guy wires and block and tackle. The successive steps in this process are shown in Plates XII, *B*, XIII, *A*, XIII, *B*, XIV, *A*, and XIV, *B*. When the hood has reached the vertical position, if the fire is a small one, the lower valves are opened and the large one closed, thereby cutting off the flame. If the flow of gas is so strong that the discharge from the 10-inch pipes may reach the top of the stack and ignite before the large valve can be closed, the 10-inch pipes are extended to a safe distance from the well before the valves are manipulated.

This principle has also been successfully used with some very large gas fires. Max Moore, an oil operator of Tulsa, Okla., applied it in 1914 to a well which, after the fire was extinguished, was found to be producing at the rate of 500 barrels of oil and 40,000,000 cubic feet of gas daily. It so happened that the well was in a river bed, the banks being about 40 feet higher than the base of the derrick.

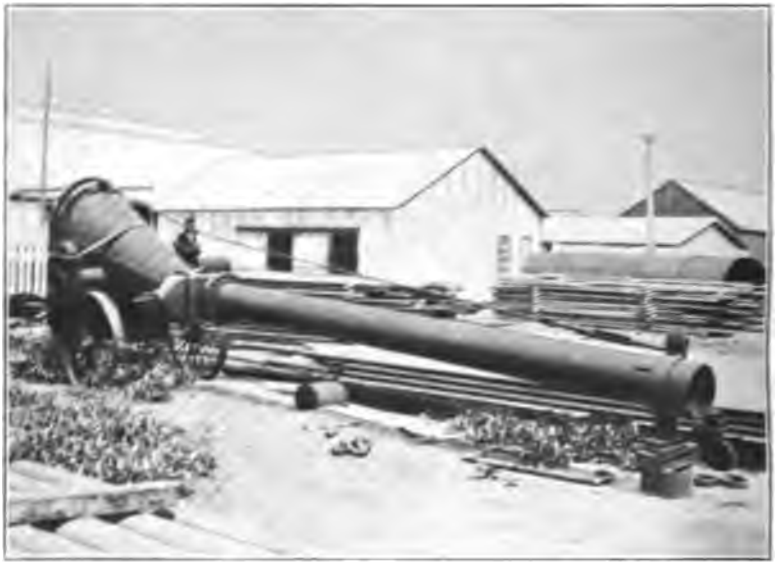
A wire cable was stretched from bank to bank in such a position as to pass just outside the stream of blazing gas. A hood made by riveting a funnel-shaped piece of pipe, the large diameter of which was about 5 feet, to an ordinary 36-inch smokestack, was hung from pulleys on the wire cable, drawn over the burning well, and placed



A. ARRANGEMENT OF GAS VENTS FROM "ALL STEEL" TANKS.



B. EXPLOSION DOOR FOR "ALL STEEL" TANK.



A. PORTABLE EXTINGUISHER FOR SMALL GAS FIRES.

B. FIRST STAGE IN PUTTING OUT GAS FIRE WITH PORTABLE EXTINGUISHER.

in position. Plate XV shows the stack nearing the fire, and Plate XVI shows it in position with the blazing column of gas issuing from the top.

The ground, rig irons, and the base of the hood were then thoroughly cooled by spraying with large quantities of water. The top of the stack was then drawn over at a considerable angle and the base was quickly removed from the well, thereby cutting off the supply of gas through the stack and extinguishing the fire.

EXTINGUISHING OIL-WELL FIRES.

Oil-well fires are often much more difficult to overcome than gas-well fires, because the well may be producing in such large quantities that the oil is not all consumed in one tapering flame, as with gas, but may flow away in a blazing mass for a considerable distance. Also, the burning oil soon heats the ground around the well so hot that no matter how often the fire may be extinguished it will immediately reignite.

Thus, no definite rules can be given for combatting fires at oil wells, as each burning well presents a problem of its own. The following descriptions of methods that have been successfully used are given, as has been stated, not with the belief that any one of these could be successfully applied to some other fire, although it might be, but with the hope that the ideas presented may, when used singly or in combination, be of service to operators in dealing with such fires in the future.

METHODS USED BY C. P. CLAYTON.

METHOD USED ON A FIRE AT A WELL IN TEXAS.

C. P. Clayton, superintendent of the Producers Oil Co., gives the following account[a] of an oil-well fire in the Humble field, Texas, which was extinguished under his supervision in February, 1916.

The well had been flowing for two days and was making about 6,000 barrels daily. It was being brought under control at the time the fire occurred, and most of the oil was discharging through a 6-inch flow line extending about 20 feet from the head of the casing. Immediately over the well, about 6 feet above the derrick floor, oil was discharging through a 2-inch opening in a 4-inch tool joint at the rate of about 100 barrels per hour. The fire was caused by an electric spark in repairing a broken power line on the derrick floor.

The well was situated in a heavily timbered country, and, as a first precaution, all timber, stumps, and other inflammable material were removed from the vicinity. Twenty-nine portable field boilers were

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[a] See also Hazlett, A. J., Ingenious method used to extinguish burning gusher at Humble: Fuel Oil Journal, vol. 7, March, 1916, pp. 78–79.

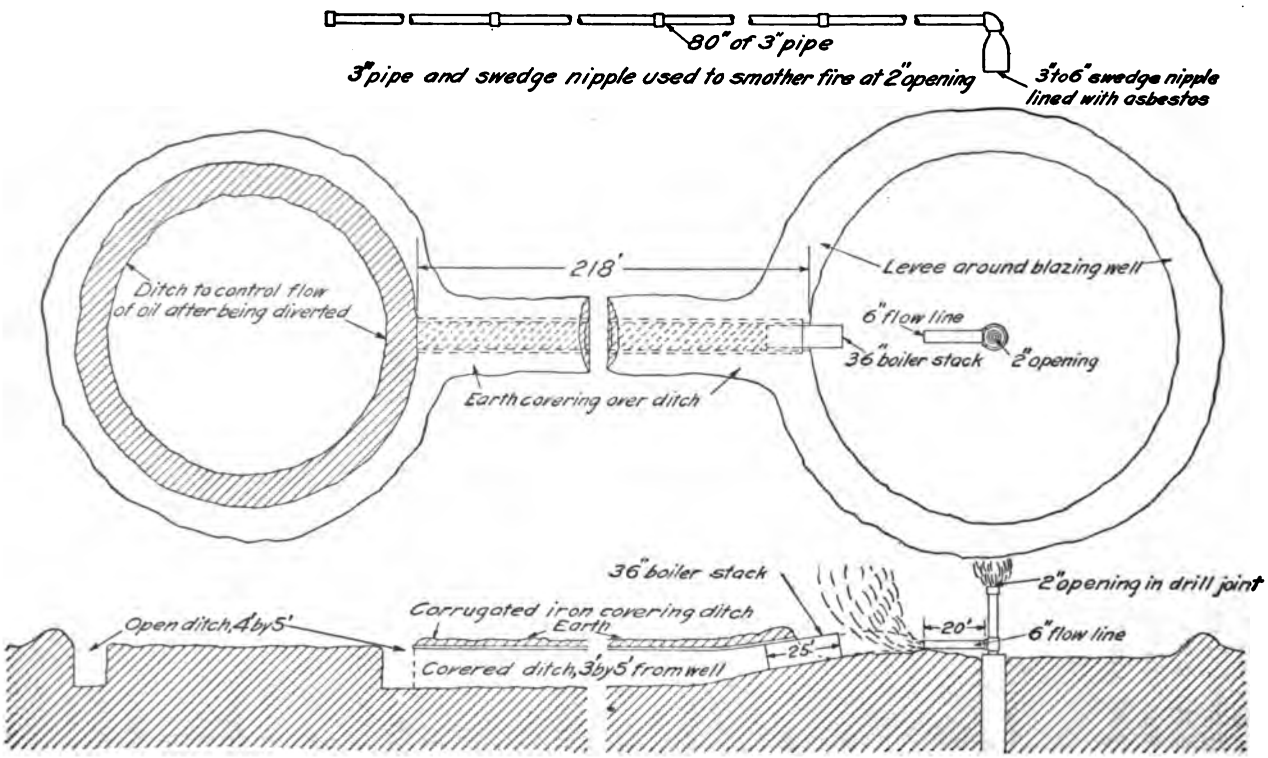
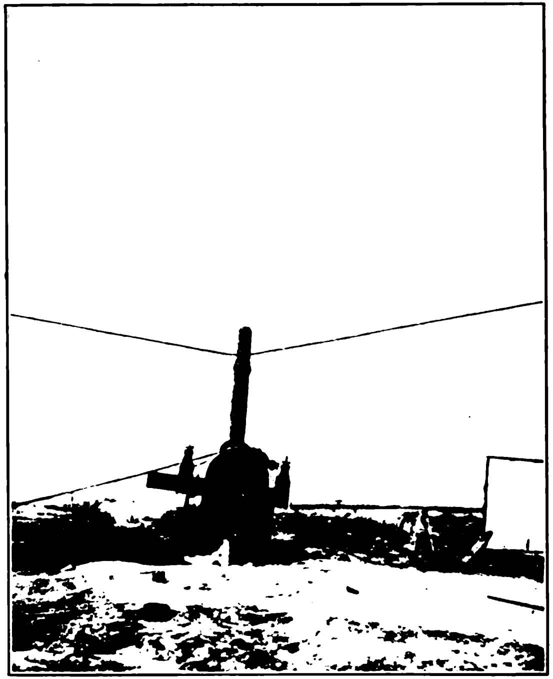
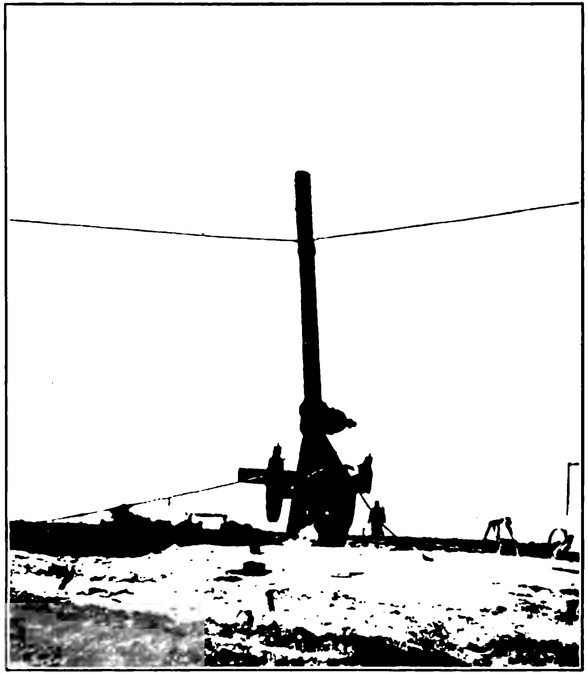

80" of 3" pipe
3" pipe and swedge nipple used to smother fire at 2" opening
3" to 6" swedge nipple lined with asbestos
218'
Ditch to control flow of oil after being diverted
Levee around blazing well
6" flow line
36" boiler stack
2" opening
Earth covering over ditch
36" boiler stack
2" opening in drill joint
Corrugated iron covering ditch
Earth
Open ditch, 4' by 5'
20'
6" flow line
25'
Covered ditch, 3' by 5' from well

FIGURE 1.—Arrangement for extinguishing an oil-well fire by forcing blaze into a boiler stack, used by C. P. Clayton.

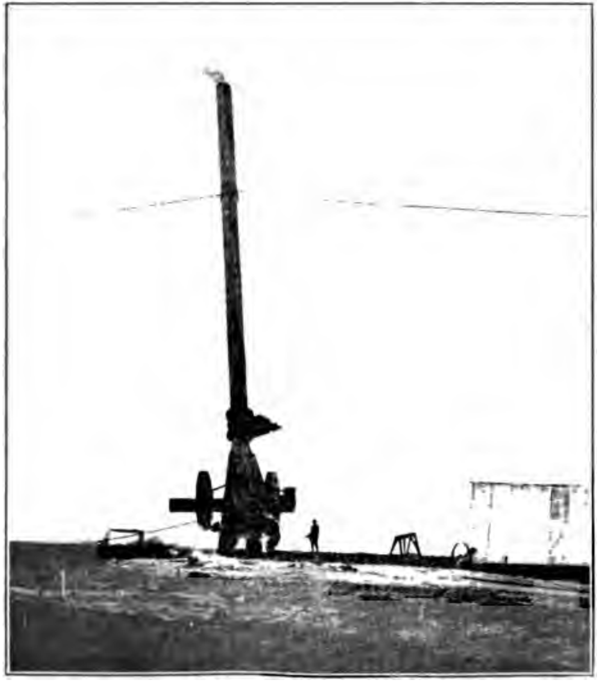
A. SECOND STAGE IN PUTTING OUT GAS FIRE.



B. THIRD STAGE.



A. FOURTH STAGE IN PUTTING OUT GAS FIRE.



B. FIFTH STAGE.

then set up, and a network of 4-inch and 2-inch steam lines laid about the well. Also, connections were made to seven boilers already installed at near-by drilling wells, and seven 2-inch water lines were laid. While this was being done a circular levee about 3 feet high was thrown up around the well, at a distance of about 50 feet from it. Opposite the flow line a ditch 218 feet long and 4 feet wide at the top, tapering to a depth of about 4 feet, was dug, which opened into a circular ditch of about the same depth and having for its outer circumference a radius of about 75 feet. This arrangement is shown graphically in figure 1. The straight ditch was then covered with corrugated iron, upon which was placed about 2 feet of earth. A piece of 36-inch smokestack was then placed in the mouth of this ditch and shoved up to within about 25 feet of the end of the flow line.

When all was in readiness, an asbestos-lined swedge nipple about 8 inches in diameter, attached to 80 feet of 3-inch pipe, was placed over the spray of burning oil discharging from the tool joint. This was done by men working from behind a corrugated iron shield, as shown in Plate XVII. At the second attempt this part of the fire was quenched and the oil discharging at this point, or the greater part of it, was made to flow through the 3-inch pipe. Immediately the steam and water was turned on the blaze at the end of the flow line (the men advancing behind shields) in such a manner as to drive the flame into the mouth of the 36-inch stack and extinguish it. The men protected their hands with gunny sacks and asbestos shields, and streams of water were continually sprayed upon them as they advanced. The preparatory work required about 17 hours; the actual quenching of the flames about 15 minutes.

METHOD USED AT A FIRE AT A WELL IN LOUISIANA.

Mr. Clayton also describes a method used by him to extinguish an oil-well fire in Caddo Parish, La., in 1911. This well burned 26 days, 2 hours and 10 minutes, and when the fire had been put out, a gage record showed a production of 48,000 barrels daily. The discharge was so great that the oil was not all consumed until some of it had flowed a distance of more than 100 feet from the well.

Thirty-six boilers were erected and the fire fought with steam for six days, but without success. At the end of this time this method was abandoned and a circular levee about 3 feet high and 200 feet in diameter was thrown up around the well, which formed a pond of blazing oil in which all the flow was consumed.

The well being on a hillside, a tunnel 4 by 6 feet by 328 feet long was driven (see fig. 2), striking the outer casing at a point 18 feet below the surface, measured to the center of the face of the tunnel. The well contained three strings of casing, 10-inch, 8-inch, and 6 inch. A split clamp was placed around the 10-inch casing, and to

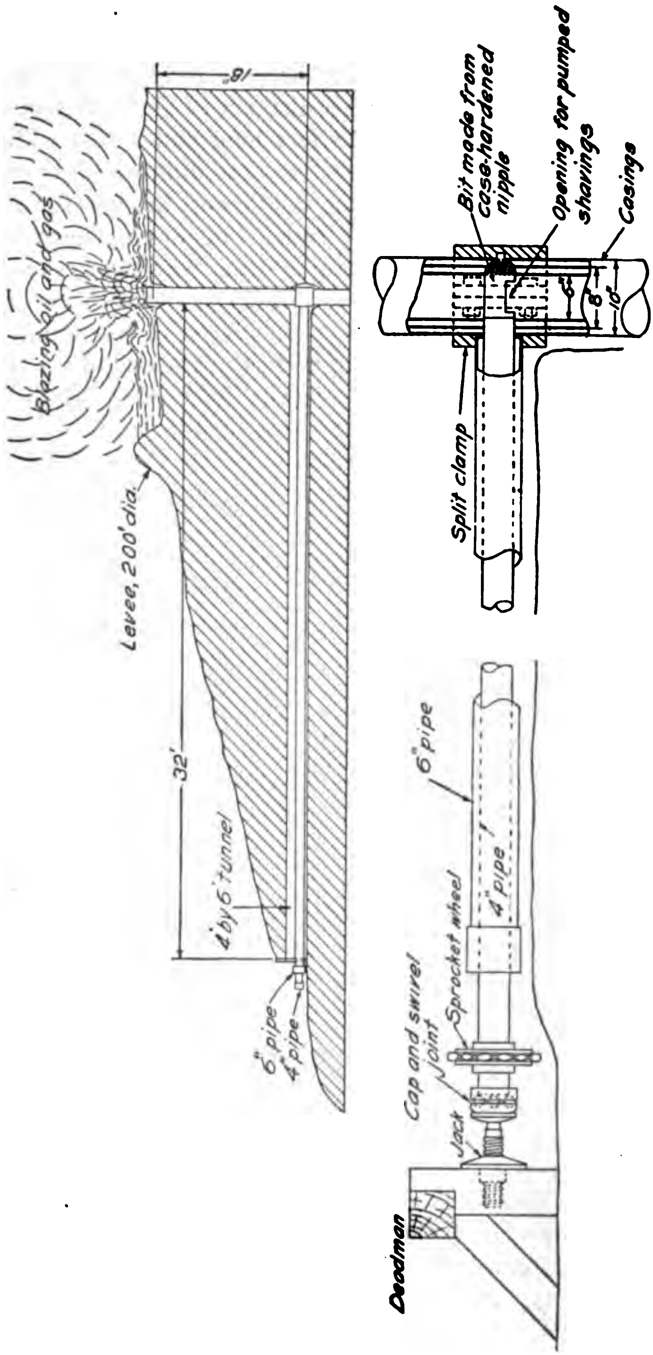
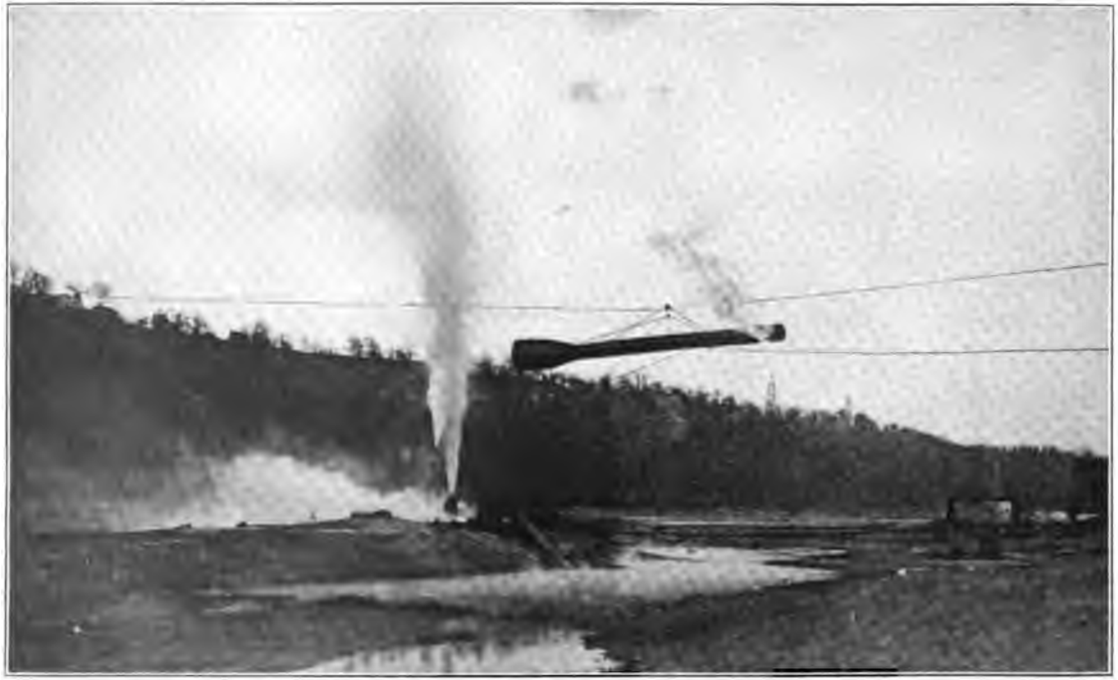
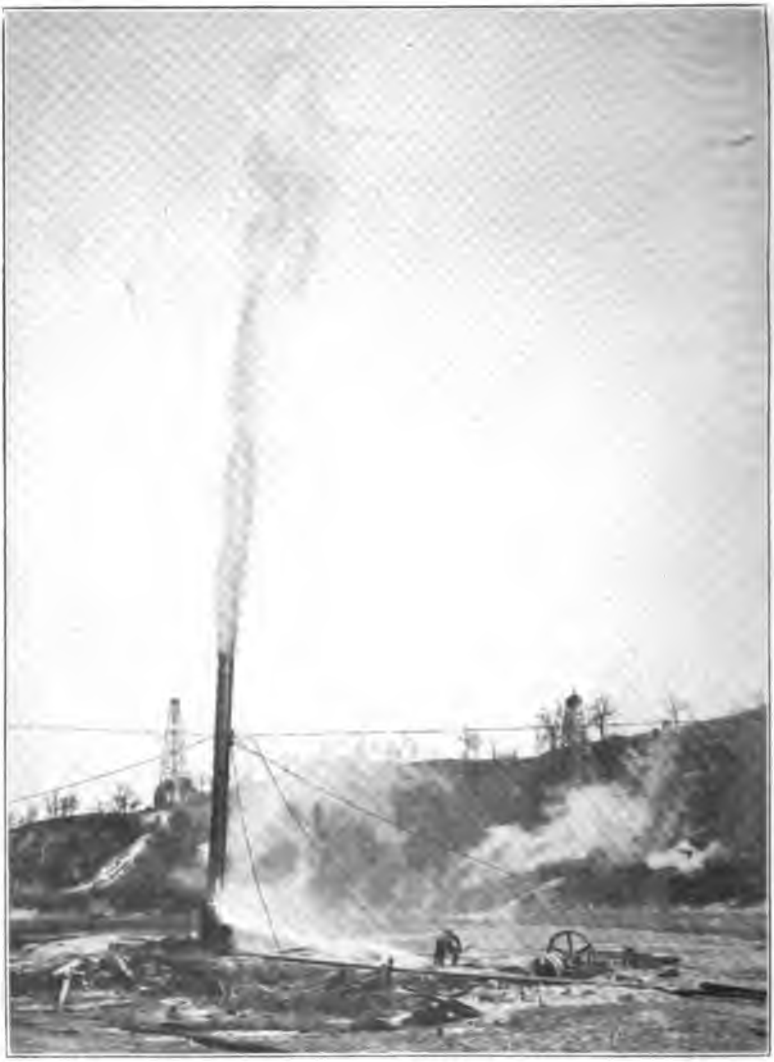


FIGURE 2.—Method of extinguishing oil-well fire by shutting off flow, used by C. P. Clayton.



PLACING A HOOD MADE FROM A 36-INCH STACK OVER A BURNING GAS WELL. FIRST STAGE.



PLACING HOOD MADE FROM 36-INCH STACK OVER A BURNING GAS WELL. SECOND STAGE.

this was attached a string of 6-inch pipe extending beyond the mouth of the tunnel. An especially constructed 4-inch bit, shown in Plate XVIII, *A*, was made from a case-hardened nipple. This bit had in one side a hole 2 inches square and was screwed onto the end of a string of 4-inch pipe; the joints of the pipe after being screwed tightly together, were also riveted. This string was placed inside the 6-inch pipe and was fitted on the outer end with a cap and a sprocket wheel to which was attached a rotary chain drive. A deadman was set opposite the end of the pipe and a jack set between it and the cap.

The chain drive was then set in motion and a hole drilled through the casings. Careful measurements were kept on the jack as the 4-inch line advanced, so that the opening in the bit would be stopped as nearly as possible in the center of the 6-inch casing. When this point was reached, the bit was turned so that the hole was "looking" down into the well. A circulating pump was connected with the 4-inch pipe and asbestos shavings forced through it under a pressure of 1,200 pounds per square inch. After this had been done for a few minutes the pipe was so turned that the hole in the bit was on top and the flow of the well was shut off. The oil on the surface, inside the levee was soon consumed and the fire extinguished from lack of fuel.

In discussing this method, Mr. Clayton states that the tunnel, where it intersected the casing, was too near the surface. He thinks the distance should have been at least 30 feet, as several of the men in placing the split clamp were almost overcome by the heat and gas fumes. Also, the tunnel was too small. Had it been 6 feet by 6 feet instead of 4 feet by 6 feet the labor of driving it and of placing the 6-inch pipe would have been greatly facilitated and the time reduced.

METHOD USED BY M. E. LOMBARDI.

M. E. Lombardi, superintendent of construction of the fuel-oil department of the Southern Pacific Railroad, describes an oil fire extinguished under his direction which occurred in the Midway oil field, California, in 1913.

The well had been drilled by the rotary system and caught fire just after it had unexpectedly started flowing, so the rotary table was still in place. After the floor of the derrick had burned the table crumpled the end of the casing, which extended several feet above the ground, until the table rested on the ground. This formed a baffle and the oil spurted out laterally with such force that before long a crater about 50 feet wide by 40 feet deep was formed.

A large number of field boilers were set up near the well and the usual method of applying steam and water resorted to. The fire was repeatedly extinguished, but the sides of the crater had become so

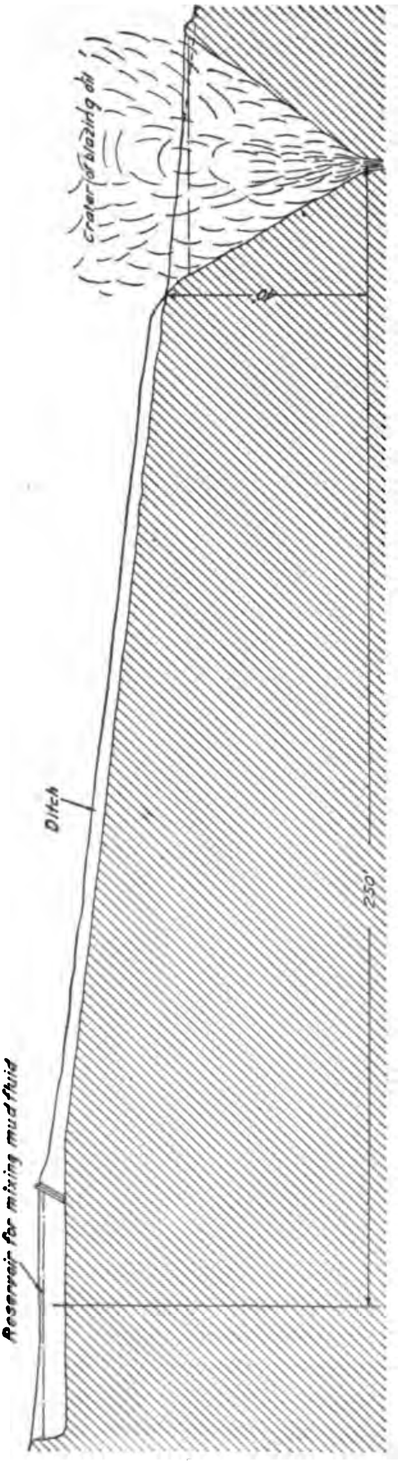


FIGURE 3.—Method of extinguishing oil-well fire with mud-fluid.

hot that the gas and the oil were reignited as soon as the blanket of steam lifted.

The well was on a hillside and the soil was a heavy clay. A pit about 50 feet in diameter and 3 feet deep was dug in the hillside about 200 feet from the well and about 30 feet above it.

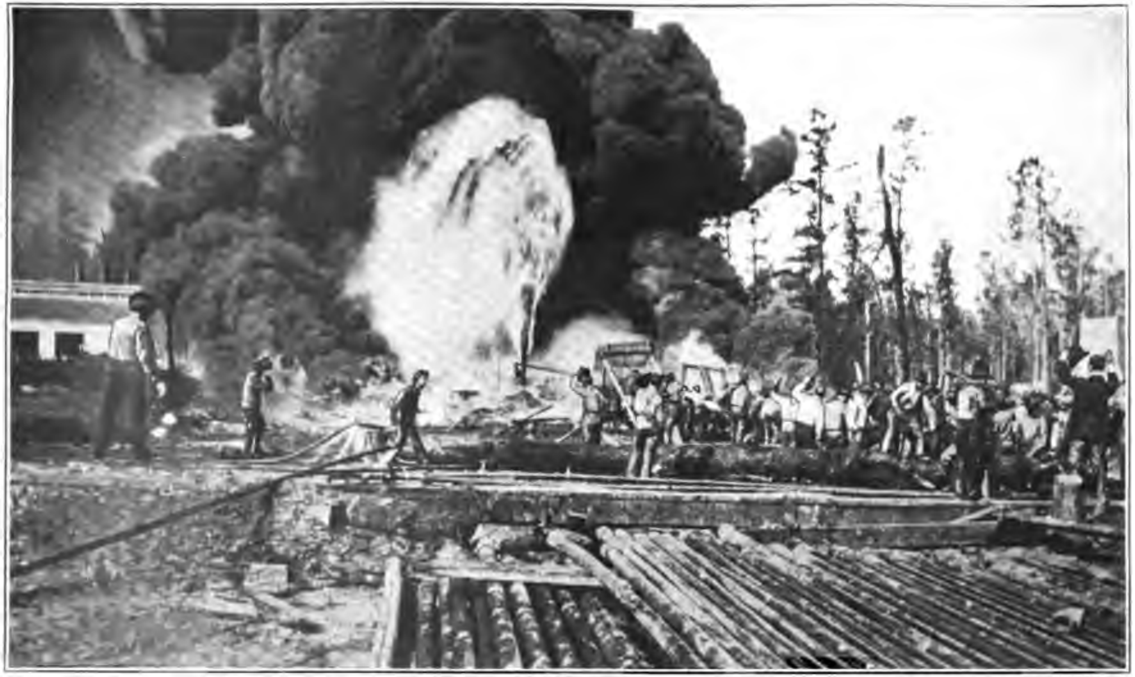
In this pit mud was mixed, fluid enough to flow freely, and the mixture was discharged into the blazing crater. The pit was filled and emptied three times in quick succession. The third application permanently extinguished the flames, the steam formed by the drying mud cooling the walls of the crater below the point where they would reignite the oil. This method is shown in figure 3.

The fire burned 16 days and, when finally conquered, the well was found to be flowing at the rate of about 1,000 barrels daily. The condition of the well after the fire had been extinguished is shown in Plate XIX.

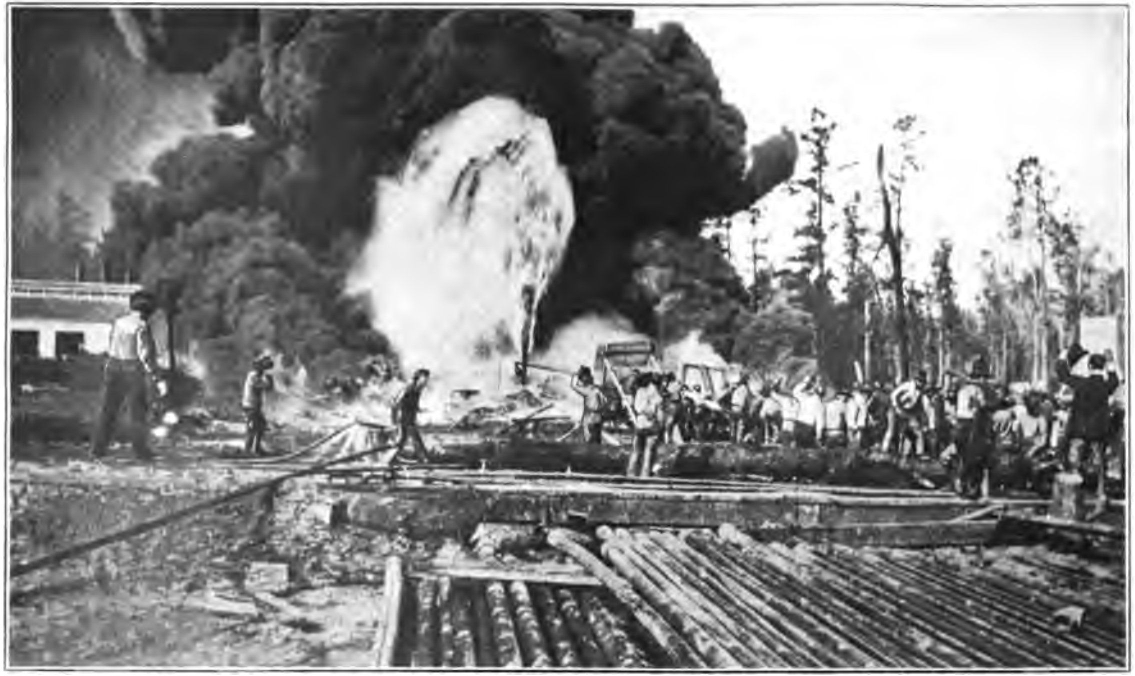
MISCELLANEOUS FIRE HAZARDS AND PREVENTIONS.

HAZARDS AND PRECAUTIONS IN AND AROUND GARAGES.

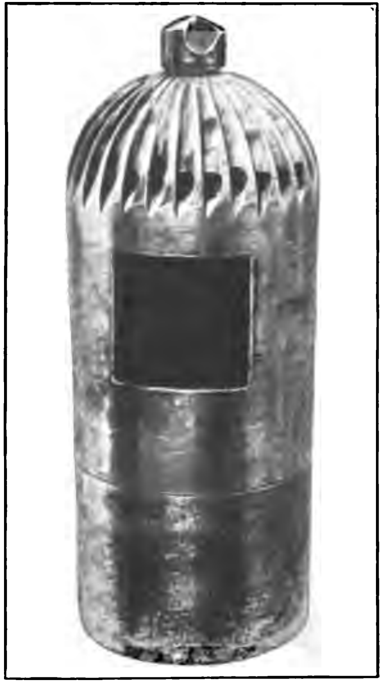
To prevent the starting of fires by frictional electricity in filtering gasoline, as described at the beginning of this paper, the use of chamois skin for straining gasoline should be done away with entirely; only fine wire gauze should be used.



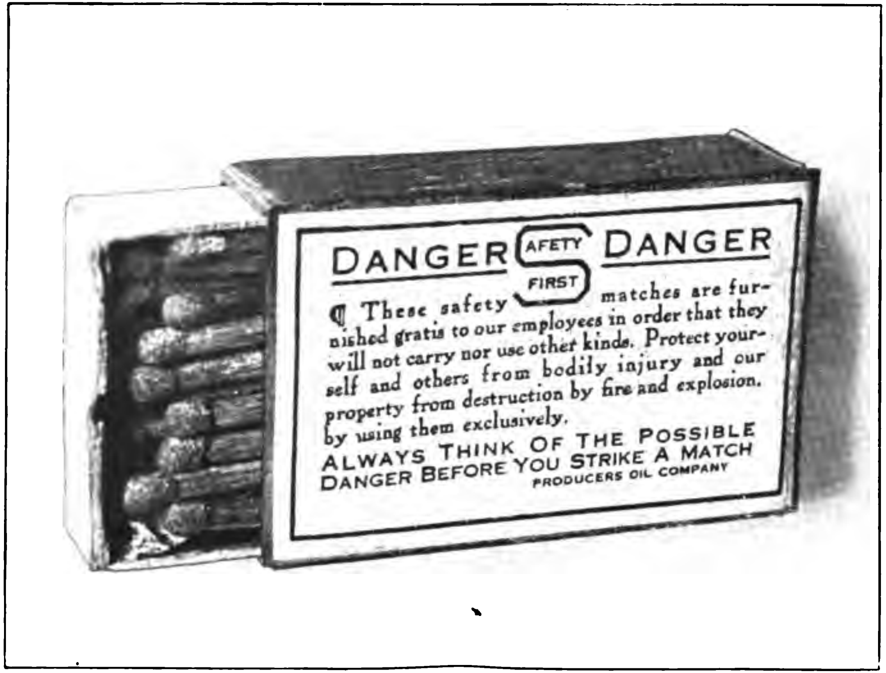
EXTINGUISHING AN OIL-WELL FIRE BY CAPPING WITH A SWEDGE NIPPLE ATTACHED TO 3-INCH PIPE.



EXTINGUISHING AN OIL-WELL FIRE BY CAPPING WITH A SWEDGE NIPPLE ATTACHED TO 3-INCH PIPE.



A. SPECIAL 4-INCH BIT MADE FROM A CASEHARDENED NIPPLE.


DANGER
SAFETY FIRST
DANGER
These safety matches are furnished gratis to our employees in order that they will not carry nor use other kinds. Protect yourself and others from bodily injury and our property from destruction by fire and explosion by using them exclusively.
ALWAYS THINK OF THE POSSIBLE DANGER BEFORE YOU STRIKE A MATCH
PRODUCERS OIL COMPANY

B. SAFETY MATCHES AND BOX, FURNISHED BY AN OIL COMPANY FOR USE BY EMPLOYEES.

In fact, many cities, especially throughout the New England States, have ordinances prohibiting the use of chamois skin for this purpose.

For drawing gasoline from pumps buckets having wooden bails should not be used, neither should wood or any other insulating material be used to support the funnel in an upright position when gasoline is being poured through it into the tank of a car. The metal of the funnel should invariably be in contact with the metal of the gasoline tank, and, if necessary, a copper chain long enough to rest on the bottom of the tank should be soldered or riveted to the bottom of the funnel spout to insure contact.

The Shell Co. of California uses spouts with copper chains soldered to them for filling its tank trucks, and insists that the chain be in contact with the bottom of the tank before the gasoline is turned on. This company also takes the precaution to ground its trucks whenever they are stabled in the garage, to make sure that any static electricity which has accumulated in them will be drawn off.

Where rubber hose is used for conveying gasoline, the discharge end should be provided with a metal spout or nozzle which has a metallic connection, such as copper wire, with the coupling on the other end. When the gasoline tank on a car is being filled, the nozzle of the hose should be in actual contact with the metal of the tank, and the container from which the gasoline is being drawn should be carefully grounded so that static electricity generated by the gasoline flowing through the hose into the tank will be drawn off to the container and grounded without the formation of a spark. These same precautions apply to gasoline filling stations for railroad cars and tanks on automobile trucks.

For putting out small gasoline fires, especially fires on the floor,[a] sawdust works surprisingly well. The sawdust floats for a time on the surface of the burning liquid, forming a blanket which excludes the air and smothers the flame. Whether the sawdust is from hard or soft wood appears to make little or no difference, and dry sawdust seems to serve as well as moist.

Mixing sodium bicarbonate with the sawdust, in about the proportion of 8 pounds of carbonate to 1 cubic foot of sawdust, increases its efficacy. This mixture has been used quite extensively in large manufacturing establishments for extinguishing fires in oils and volatile liquids. In large tanks, especially of gasoline, sawdust is not a success because it will sink before the whole surface can be covered.

Where sawdust and bicarbonate are used, the mixture should be stored in conveniently placed cans of suitable sizes, provided with a wide door in the side near the bottom, and a scoop shovel, so that the material can be spread rapidly when required. Where these containers are not too large, say 3 feet square, there is no danger from spontaneous combustion. However, sawdust alone, if it becomes soaked with oil drippings,[a] will soon begin to smolder and in time will burst into flames as oily waste will do.

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[a] Barrier, E. A., Extinguishing fires in oils and volatile liquids in paint and varnish manufactories: American Paint Jour., April 9, 1917.

Carbon tetrachloride, as put up by some manufacturers in portable containers of the force pump type, is useful in fighting small fires. Carbon tetrachloride is a water-white liquid that is noninflammable and is a poor conductor of electricity; it has a specific gravity of 1.632 at 32° F., and is readily miscible with oils. When put up in commercial extinguishers it is usually mixed with some agent such as chloroform, or acetylene chloride, which reduces the freezing point to 20 or 30 degrees below zero.

The specific gravity of its vapor is about 5½ times that of air. When applied to burning oil, carbon terachloride tends to extinguish the fire in two ways, by mixing with the oil, thereby diluting it, and also by forming a blanket of noncombustible gas which excludes the air. When it is heated the unconsumed particles of carbon cause a dense black smoke. The fumes are pungent, but brief exposure to them does not cause permanent injury. For best results, carbon tetrachloride should be sprayed on the edges of the fire and not into the body of it. Where a relatively large fire is to be overcome the efficacy of carbon tetrachloride depends largely on the skill of the user.

"Frothy-mixture" solutions, such as those described on page 11 and 16, when put up in portable containers of convenient sizes also are useful in extinguishing small fires about garages. The commonly used acid and soda type of extinguisher is better adapted to fires in woodwork than in oil because of the large excess of water discharged with the gas bubbles, and the flimsy nature of the bubbles.

Wherever possible, filling stations for automobiles should be in the open and not inside of buildings, as in any event highly inflammable gases will arise from the opening of the gasoline reservoir when the automobile tank is being filled. Where this work must be performed inside of buildings, proper ventilation should be maintained at all times—in winter as well as in summer.

Smoking about garages and filling stations should be strictly prohibited, and this rule should apply to the customer having his car filled, as well as to the attendant. All flame lights about the car should be extinguished and the engines stopped before the car is served. No electric switches or fuses should be placed in the filling room.

Buildings for garages should be of fireproof construction throughout. Machine and repair shops run in connection with them should

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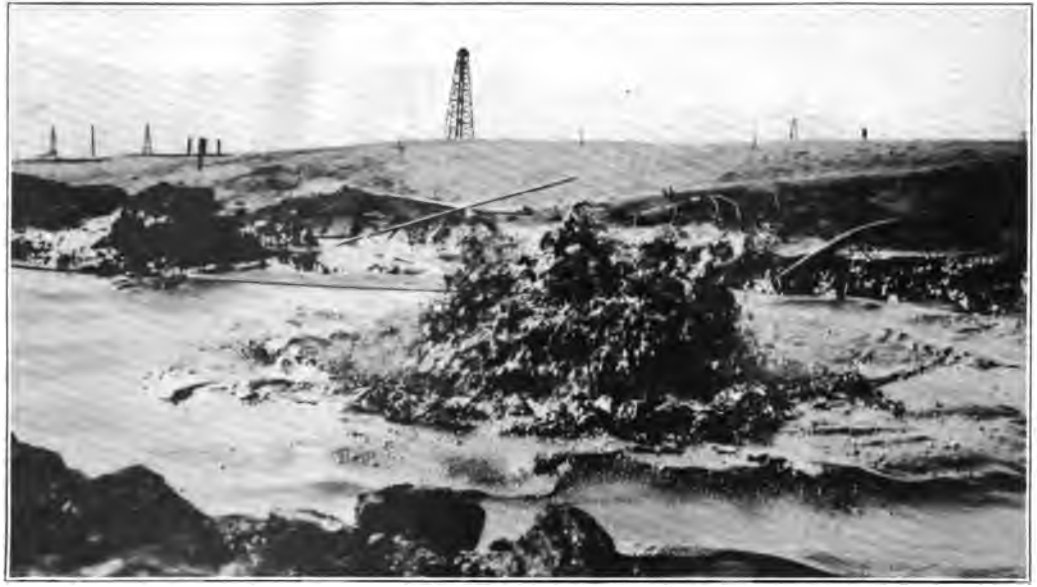
[a] Burrell, G. A., Hazards in handling gasoline: Tech. Paper 127, Bureau of Mines, 5, p. 6.



CONDITION OF FLOWING OIL WELL, WHICH HAD BURNED FOR 16 DAYS, AFTER FIRE WAS EXTINGUISHED.

be in separate buildings, or, if in the same building, should be protected with a fire wall and automatic fire doors. Electric lights, only, should be used and these should be properly protected with wire guards. Open lights or fires of any kind should never be permitted in that part of the garage where there are automobiles. Heating of the buildings should be by steam or hot water.

The buildings should be cleaned thoroughly every day, and all rubbish and other waste material should be gathered up and destroyed. Covered metal receptacles should be provided for all discarded oily waste, as this material when compressed in a lump and allowed to lie unmolested may spontaneously ignite in a short time.

All gasoline storage tanks should be placed underground, preferably outside the building. Gasoline should be drawn from these by pumps; in no case should a gravity flow be arranged. All liquid in the pipes should drain back into the storage tank when the pump is stopped. No cans or tanks of gasoline should be allowed on the floor of the garage other than necessary portable wheeled tanks, none of which should exceed 60-gallon capacity.

The use of gasoline for cleaning any part of an automobile should be prohibited except in a special room, thoroughly ventilated and used for this purpose only. Where such cleaning is done out of doors, it should not be performed closer than 15 feet from any door or opening in the buildings.

Garage waste basins, sinks, or floor drains should never be directly connected with any house drainage or sewer system. Vapors arising from light hydrocarbon compounds, when mixed in proper proportions with air, are highly explosive, and when confined in a sewer system are a constant source of danger.[a] All grease, oil, and inflammable liquids should be removed from waste water by traps or separators before the water is permitted to enter the sewers.

"No smoking" signs should be posted in prominent places about the premises, as well as other signs explaining the dangers involved in handling the pumps or other apparatus, and giving safe methods for operation.

HAZARDS AND PRECAUTIONS ABOUT TANK FARMS AND REFINERIES.

TANK CONSTRUCTION.

As has already been stated, all tanks about tank farms and refineries should be of steel construction throughout with gas-tight roofs on steel roof supports. Tank stairways should also be of steel. Gage holes should be fitted with tight iron caps, which can be screwed down and will prevent the escape of gas. Swing pipe lines should pass through stuffing boxes of some such type as that described in Bulletin 155.[a] These boxes should be kept at all times in gas-tight condition. A vent pipe of proper size should lead from a point near the apex of the roof down the side of the tank, and thence underground on a true gradient to a point outside the fire levee, where it should terminate in a vertical stand pipe at least 20 feet high. This pipe should be so laid that the lowest point is at the bottom of the stand pipe, where a small drain should be provided to remove any possible accumulation of water or condensate from the gases. A tee having each run closed with a fine meshed wire gauze, should be " bull-headed " into the top of the stand pipe.

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[a] Burrell, G. A., and Boyd, H. T., Quantity of gasoline necessary to produce explosive conditions in sewers: Tech. Paper 117, Bureau of Mines, 1916, p. 16.

As has been previously mentioned, an arrangement whereby gases from vent pipes would be passed through an absorber or condenser or otherwise profitably utilized, would be far better than releasing the gases to the atmosphere.

PIPE LINES.

Pipe lines connecting storage tanks should be underground for at least part of the distance, or otherwise thoroughly grounded. The writer has been unable to find any instance where a tank has been ignited from lightning following exposed piping, but such a condition might easily exist. Therefore it is a wise precaution to bury the pipes in the ground or, in some localities, even place insulating joints in them.

LOADING STATIONS.

Fires are said to have been caused at loading stations, where ships discharge or take on oil, through electrical sparks formed when connection to the ship was being made or broken. The writer has been unable to find any instance where the origin of the fire was actually traced to this cause, but it is quite possible that fires could be started in this way, especially where the pipe lines leading from the wharves pass near power lines, electric railways, etc. Most companies ground their loading pipes at the wharf. This is done by soldering a copper wire to the pipe; the other end of the wire is attached to a copper plate, which is sunk in the mud beneath the water.

SUCTION PIPES.

All tanks should be provided with large suction pipes so that in case of fire as much oil as possible can be removed to other tanks or reservoirs. Tanks containing crude oil should also be provided with bottom drains, connected with piping discharging at a safe distance from the tank. As the tank burns, these drains should be kept open to remove the water that settles from the oil. This precaution is very important, as a small amount of water in the bottom of a tank when converted into steam will cause the tank to boil over violently and the blazing froth may be carried by the wind along the ground and over the top of the fire levees, spreading the fire to other tanks or structures.

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[a] Bowie, C. P., Oil storage tanks and reservoirs, with a brief discussion of losses of oil in storage and methods of prevention: Bull. 155, Bureau of Mines, 1918, p. 17.

LEVEES.

Where possible, tanks should be surrounded with a dike or levee of sufficient height to hold 1½ times the capacity of the tank. Each tank having a capacity of 5,000 barrels or more, which is used for the storage of oil, should be placed at least 300 feet, measured from shell to shell, from any other tank, and at least 150 feet from derricks, buildings, or other inflammable structures.

For a distance of at least 150 feet from the tank the ground should be kept free from weeds, grass, and rubbish.

LIGHTNING RODS.

The writer has made many inquiries among operators regarding the use of lightning rods, but has been unable to find any instances where there is proof that their use has prevented a tank or other structure from being struck. The fact that they present a line of least resistance for electricity to pass from a rain cloud into the ground might prove a serious menace to the structure upon which they are placed because of the enormous voltage they might be required to carry.

PRECAUTIONS ABOUT OIL STILLS, ETC.

Oil stills, heat exchangers, agitators, dephlegmators, etc., should preferably be placed in the open. Where protection from the elements is necessary, roofs and roof supports should be of fireproof materials throughout.

Concrete or brick fire walls of reasonable height (say 3 feet 6 inches) should be constructed at various points about the stills and other apparatus, dividing the entire installation into small units where possible fires can be confined until they are extinguished. In the opinion of the writer this provision is important and should be incorporated into the design of every refinery constructed.

RUN-DOWN TANKS, ETC.

Storage tanks, run-down tanks, stock tanks, agitators, and other closed containers should be equipped with steam smothering lines which will admit steam into the top of the container on a moment's notice. On the approach of a thunderstorm, steam should be turned into the tanks, and the space between the surface of the oil and the roof of the tank kept filled with steam until the storm has passed.

POSITION OF TRANSFORMERS.

Where electric power is used for operating any part of a refinery the transformers should be at a safe distance from all sources of inflammable gases. This will usually mean placing the transformers outside of the refinery. Great care should also be exercised in placing switches and motors, as sparks from making and breaking of circuits have often been the cause of disastrous fires.

SMOKING.

Forbid smoking about the plant and see that the rule is enforced. Post "No smoking" signs in all prominent places. Provide a room where the men can smoke after the lunch hour. This room should be away from the other structures and should be of fireproof construction throughout.

MATCHES.

Forbid the use of "strike anywhere" matches. Permit no matches to be used about the plant except those that strike only on the box in which they are furnished. The Producers' Oil Co. furnishes its employees with safety matches, free of charge, put up in boxes having a warning printed on the outside. One of these boxes is shown in Plate XVIII, *B*.

CHANGE ROOMS.

Provide well-ventilated metal lockers for the employees' clothing. These should be installed in a fireproof building such as the smoking room. Never permit employees to dry wet clothing on steam radiators; drying clothing in this way has caused many a fire. Wet garments should be hung up to dry by circulation of air. Never permit employees to hang wet garments on electric wires.

FIRE BRIGADES.

Tank farms and refineries should equip and maintain a fire brigade, the members being employees about the plant. If the plant force is too small for forming such an organization, the force should be combined with that of some adjacent plant or factory. The organization of the brigade should be along the lines of a city fire company. In organizing and equipping it, probably no better advice could be obtained than that of the fire chief in the city or town where the plant is situated or in the nearest adjacent city.

WATER SYSTEMS.

A high-pressure water system should be maintained, and fire hydrants should be placed at convenient positions about the plant, especially in the vicinity of wooden structures. The hydrants should be of standard construction and should have at least two 2½-inch outlets with separate valves. The threads on the outlets should correspond exactly to those on the hose of the nearest city fire department. If there is no public fire department then the threads should correspond to those on the hose coupling of the nearest factory or refinery. This is very important.

Caps to hydrant outlets should be kept well oiled and not permitted to get jammed. All hydrants should be tested at least once a year. In cold localities this should be done in the summer time.

Cotton, rubber-lined hose should be tested once a year with water under a pressure of 100 to 125 pounds. Also, water should be run through it at least three times a year to preserve the lining. Unlined cotton hose should not be tested with water; such hose rots quickly from moisture and should be kept as dry as possible at all times. Extra hose should preferably be hung up or placed on an inclined plane. Where this can not be done, it should be well dried, then loosely rolled, and laid flat on a shelf. Hose that has been used with salt water should be thoroughly cleaned with fresh water before being dried.

HAZARDS AND PRECAUTIONS ABOUT DRILLING AND PRODUCING WELLS.

Many fires about drilling wells have been started by having the forge inside of the derrick, also by having an open fire in the derrick in cold weather. Neither of these practices should be permitted. Tools should be dressed at a reasonable distance from the rig, or, better still, at some central station. However, in "wild cat" work sending tools to a central station is seldom feasible.

Gas fires about oil wells being drilled are in most instances wholly inexcusable, as permitting gas to escape unchecked from the well is a wasteful and, as a rule, unnecessary procedure. The gas should be shut in as the drilling proceeds.[a]

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[a] Lewis, J. O., and McMurray, W. F., The use of mud-laden fluid in oil and gas wells: Bull. 134, Bureau of Mines, 1916.

POSITION OF BOILERS.

Boilers should be so placed that they will not be a source of danger if a flowing well is brought in, or if flow tanks or stock tanks are overtopped. In some districts having prevailing winds, as along sea coasts, it may be feasible to place the boilers on the windward side of the well so that the greater part of the time the well gases will be blown away from and not toward the boilers. It is well to have wire gauze protectors over the tops of the boiler stacks, even at plants where oil is used for firing the boilers.

LIGHTS FOR RIGS.

Drilling rigs should be lighted, where possible, with electric lights and not with open torches. At producing wells, only electric lights should be used; neither these nor the wiring should be placed on the derrick, but rather on a pole several feet from it, and the bulbs should be protected with wire guards. If the well is flowing or producing much gas, it is better to have the lights in a cluster on a pole 50 feet or farther from the well.

LUBRICATION OF BEARINGS.

All bearings about the machinery, especially the bull-wheel, calf-wheel, and sheave-wheel bearings, should be kept thoroughly oiled. Frayed bull ropes should be discarded and replaced with new ropes, as any rubbing or slipping of the rope may create heat sufficient to ignite gas issuing from the well. The bull-wheel brake should be provided with a properly installed water spray and this should be used before, not after, the bull wheel begins to smoke.

STATIC ELECTRICITY.

Static electricity generated by the band-wheel belt, or other moving parts of the machinery, has often caused fires. J. C. McCue, superintendent of the Producers Oil Co., cites an instance where he watched static electricity sparking between the band-wheel belt and a wet board, one end of which was buried in the ground, for two hours or more, until the board burst into flame. Mr. McCue takes the precaution to see that all machinery parts are grounded and that belts are provided with properly constructed copper brushes attached to pipe; the pipe being grounded.

An instance has been brought to the attention of the writer where a gas well in process of being drilled was ignited three times before the cause was found to be static electricity generated at the band wheel, passing through the shaft of that wheel into the sand reel and along the sand line to the bottom of the bailer, where it formed a spark in discharging from the bailer into the top of the casing, as the bailer was being lowered into the well.

SHOOTING WELLS AND BLOW-OUTS.

Fires have been caused in shooting wells, and in striking gas pockets while drilling, by rocks being blown forcibly against the sheave wheels or rig irons.

However, in the majority of blow-outs the fire, if there is one, will be caused by gas or oil coming in contact with the boiler fires. Sparks produced by rocks striking against the rig iron will not ignite the column of gas or oil unless the spark happens to be formed on the outside of the column where there is sufficient air for combustion.

ALLOWING DRILLING CABLE TO SLIP THROUGH CLAMPS.

Numerous instances are on record where fires have been started about drilling wells, which were making a small amount of gas, through permitting the wire drilling cable to slip through the drilling clamps when "hitching" the tools over. This is bad practice, for not only is the fire hazard increased but the cable and equipment may be damaged or the driller and his assistants injured.

PRECAUTIONS ABOUT WELLS WHERE COMPRESSED AIR IS USED.

In using compressed air for increasing the recovery of oil from wells, it is stated[a] that "air-gas" used in gas engines has been ignited and the flame traveled back in the pipes to the well. This can be prevented by inserting at the proper point in the gas line a section of larger pipe filled with small iron tubes or rods that will smother the flames by cooling, much the same as the gauze in a safety lamp prevents gases being ignited by the flame of the lamp. Sometimes oil will flow from a well by the compressed air, and fires are reported to have been caused by oil getting into the air-gas and reaching the gas engines. This can be prevented by installing a trap in the gas line, through which the gas must pass before reaching the engine.

GAS SCRUBBERS.

Where natural gas is taken directly from a well and used to drive engines, fires have been started by sand in the gas striking the sides of the exhaust pipe and causing sparks. This danger can be eliminated by passing the gas through a "scrubber" before it is admitted to the engine. Ordinary tubular boilers are often used as scrubbers. Valves should be placed in such gas lines at safe distances from the engines so that if the exhaust catches fire the supply can be quickly shut off.

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[a] Lewis, J. O., Methods for increasing the recovery from oil sands: Bull. 148, Bureau of Mines, 1917, pp. 82–83.

ACCUMULATION OF GAS IN RAVINES.

Disastrous fires have been caused by gas from flowing oil wells accumulating in ravines and low places in the vicinity of the wells and there being ignited, the fire striking back to the well along the stratum of gas.

The danger of allowing gas to accumulate in this manner is exemplified by a serious accident[a] near Drumright, Okla., in May, *1915*, although the fire did not strike back to the well. It was a calm, humid evening. An automobile, with a man and two women in it, was passing near some flowing oil wells. As the car crossed a culvert over a small creek, there was a terrific explosion. The women were so badly burned that they died within a few hours. The man was also badly burned. Investigation showed that a body of gas had accumulated along the creek bed, and the head lights of the automobile had ignited this gas. The flash was seen by people several miles from the scene of the accident.

GAS TRAPS.

As soon as possible after a well has been brought in, suitable gas traps should be installed and the gas condensed or otherwise utilized. Permitting gas to escape into the atmosphere is a needless waste, in addition to being a menace to life and property.

ARRANGEMENT OF VALVES FOR FLOWING WELLS.

Many companies equip their flowing wells with an arrangement of valves, as shown in figure 4. The lower valve has an extension stem, made from 1-inch pipe, of such length that the valve can be operated at 50 or 75 feet from the derrick. If the derrick, or the oil issuing from the flow lines should catch fire, this valve can be closed and the oil shut off until the fire is extinguished.

MATCHES.

"Strike anywhere" matches are responsible for many fires. Their use should be strictly prohibited about producing wells and drilling wells from which gas is flowing. As has been previously pointed out, only matches that strike on the box in which they are furnished

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[a] News item, Two women lose their lives; man with them badly burned: Oil and Gas Jour., vol. 13, May 20, 1915, p. 23.

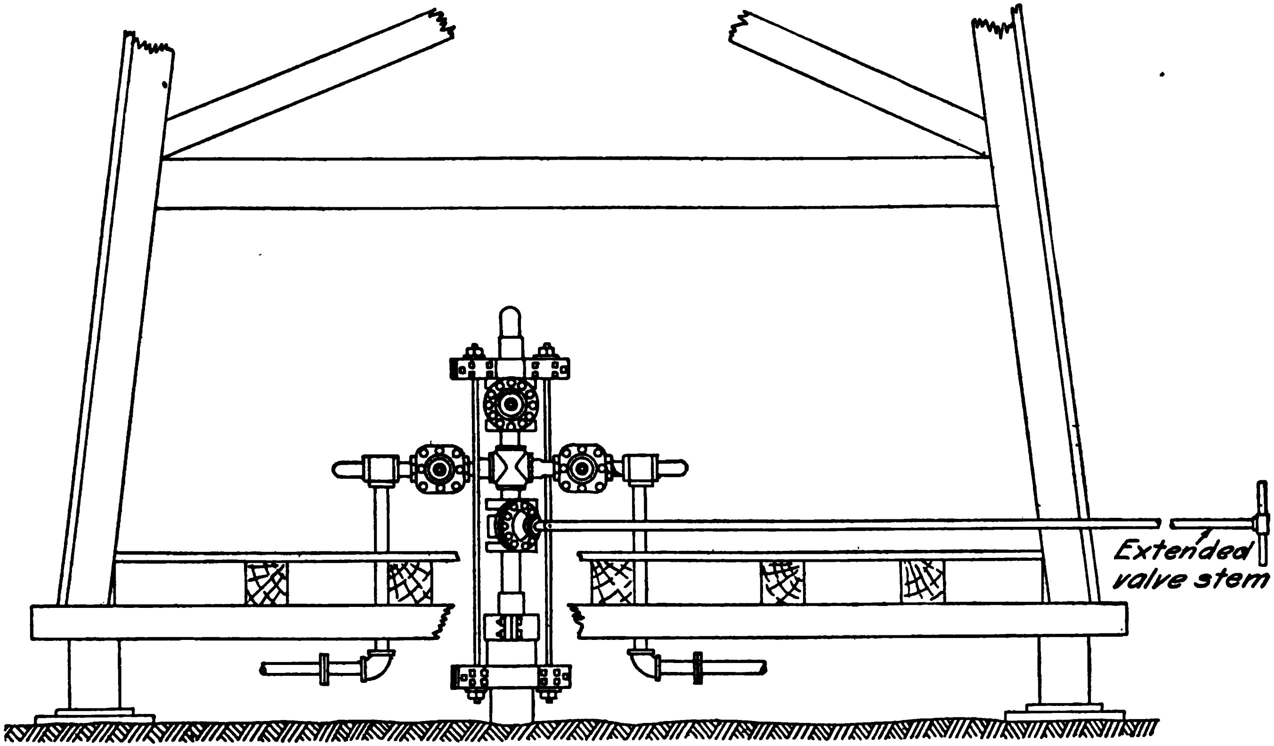

FIGURE 4.—Arrangement of valves for flowing wells.

should be permitted. "No smoking" signs should be posted about such wells, and the rule should be enforced.

INSURANCE RATES.

If the precautions outlined in this paper are put in practice by operators, the result will be a material reduction in fire hazards, which should reduce fire insurance rates. Just what the amounts of the reduction in insurance rates will be depends on the nature of the risks, and the particular underwriting company covering the district.

It is recommended that operators, when laying out new work or providing fire-fighting apparatus for structures already in place, consult the underwriters covering their particular district and adjust their installations, as far as possible, to conform with the requirements of these companies, in order that a maximum reduction in rates (if insurance is to be carried) may be obtained.

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Bulletin 173

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

MANGANESE

USES, PREPARATION, MINING COSTS AND THE PRODUCTION OF FERRO-ALLOYS

BY

C. M. WELD and OTHERS

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PREFACE.

The history of the domestic production of manganese ores and alloys during the war, in common with that of several other materials equally essential for war purposes, is of interest because of its showing how a hitherto latent industry responded quickly to the spur of necessity.

In the past the supplies of manganese ores used in this country have come largely from Russia and India. More recently imports from Brazil began to assume importance, but up to 1914 they were still a relatively minor factor. Also, small amounts came from Cuba and Central America. The domestic output, however, was practically negligible.

The European war soon shut off Russian sources, and receipts from India declined. The domestic output became somewhat larger, but the deficit was principally made up by greatly increased imports from Brazil.

When this country entered the war, the manganese situation became acute. Manganese being essential in the manufacture of steel, the insuring of an adequate supply was imperative. On account of the increasing difficulty of obtaining foreign ores, through uncertain output and shortage of shipping, it was obvious that this country must turn to its own deposits. In July, 1917, the war minerals committee was established, and surveys of the domestic manganese situation were begun. By the end of the year the domestic output had reached hitherto unprecedented figures.

Early in 1918 the need of diverting every available ship to the transportation of troops and military supplies made necessary the reducing to the minimum of all imports of materials that could be produced in this country. The manganese situation was studied most thoroughly by various governmental bodies, in cooperation with the ferro-alloys subcommittee of the American Iron and Steel Institute, and a program was formulated for reducing overseas imports to the minimum figure consistent with the production of an adequate supply of steel. To stimulate domestic production, a higher price schedule was announced by the American Iron and Steel Institute, with the approval of the War Industries Board. At the same time the market for leaner domestic ores was broadened by lowering the standard grade for manganese alloys.

As a result, had the war continued through the year 1918, there would have been produced more than 300,000 tons of domestic ores containing not less than 35 per cent metallic manganese, with larger quantities than ever before of the lower grade ores suitable for the manufacture of spiegeleisen. In addition, alloys of domestic manufacture of the alloys had entirely replaced those formerly imported. In November, 1918, when hostilities ceased, there was nearly a year's supply of ore and alloys on hand. As a result, the buying stopped almost at once and production fell rapidly. Mines were shut down and furnaces were blown out or used for other products.

Various proposed measures for stabilizing the industry were considered, but none seemed to meet the situation. In 1919 the war minerals relief bill was passed, establishing a commission for investigating losses incurred in the production of manganese and certain other minerals and for reimbursement where the investments were due to representations of the Government.

During the war the engineers connected with the war minerals investigations of the Bureau of Mines kept in touch with developments in the industry. Wherever possible advice and help in mining and milling problems were given by correspondence or in person. Metallurgical problems were carefully studied, not only those bearing on the manufacture of the alloys, but those relating to the use of the alloys in the manufacture of steel. The chief end kept in view was to eliminate waste and to widen and popularize the use of the leaner domestic ores.

In order to supplement the field work and make public the results of its investigations, the Bureau of Mines published a series of mimeographed bulletins dealing with those phases of the industry, which were studied in some detail. The object of this report is to present these papers in more permanent form, in the hope that the information may be of present or future value to the industry.

J. E. Spurr,
Executive, War Minerals Investigations.

MANGANESE USES: PREPARATION, MINING COSTS, AND THE PRODUCTION OF FERRO-ALLOYS.

By C. M. WELD and Others.

INTRODUCTION.

During the past two years the Bureau of Mines has issued a series of mimeographed reports giving the results of research work and experiments conducted as part of its war minerals investigations.

In this bulletin the reports on manganese are presented, which range in scope from the beneficiation of the ore to the utilization of the metal. The bulletin is in eleven chapters, each comprising a separate report, arranged in the order given below.

1. General information regarding manganese, by C. M. Weld.
2. Uses of manganese other than in steel making, by W. C. Phalen.
3. Problems involved in the concentration and utilization of domestic low-grade manganese ores, by Edmund Newton.
4. Preparation of manganese ore, by W. R. Crane.
5. Leaching of manganese ores with sulphur dioxide, by C. E. Van Barneveld.
6. The Jones process for concentrating manganese ores; results of laboratory investigations, by Peter Christianson and W. H. Hunter.
7. Cost of producing ferromanganese ores, by C. M. Weld and W. R. Crane.
8. Production of manganese alloys in the blast furnace, by P. H. Royster.
9. National importance of allocating low-ash coke to the manganese-alloy furnaces, by P. H. Royster.
10. Electric smelting of domestic manganese ores, by H. W. Gillett and C. E. Williams.
11. Use of manganese alloys in open-hearth steel practice, by Samuel L. Hoyt.

As each chapter was originally prepared as a unit, there is necessarily some duplication and overlapping. With certain exceptions, however, each paper is reproduced in practically its original form.

In chapter 1, covering certain general phases of the industry, such as uses, specifications, prices, and statistics, the subject matter has been somewhat modified in view of changes in conditions.

About 95 per cent of the manganese consumed in the United States goes into the manufacture of steel. The remaining 5 per cent is used in a number of minor industries, the chief of which is the manufacture of dry cells. These minor uses are described in chapter 2.

No discussion of the geology of manganese ores is presented, but a short bibliography on the subject is appended. With few exceptions the manganese deposits of the United States are irregular, pockety, and uncertain. Largely for this reason, mining methods are crude and hardly warrant descriptions.

The concentration of manganese ores, however, is an important problem and involves some details not common to other ores. Chapter 3 discusses the concentration and utilization of low-grade ores, notes the relationship of concentration to metallurgical practices, outlines concentrating processes, and treats of commercial problems.

In general, as regards concentration methods, manganese ores may be divided into two groups, as follows: (1) Ores permitting mechanical separation of the manganese minerals and the gangue, and (2) ores in which the manganese minerals and the gangues are so intimately associated that separation requires some hydrometallurgical or pyrometallurgical process. Heretofore, attention has been confined almost wholly to mechanical separation, chiefly by gravity. In fact, so far as known, all production of concentrates has been by wet gravity methods, although during the war some companies investigated the commercial possibilities of magnetic separation and reported favorable results, the construction of one magnetic concentration mill being started. The usual wet gravity methods are described in chapter 4.

During the war, the Bureau of Mines undertook investigations of methods applicable to ores of group 2. A hydrometallurgical process involving leaching with sulphur dioxide was studied at the mining experiment station of the bureau at Tucson, Ariz. The results of this work, presented in chapter 5, indicate that the process is metallurgically feasible but the cost makes it unattractive.

The results of an investigation of a pyrometallurgical process known as the Jones process are presented in chapter 6. The process, which is still in the experimental stage, was found to be metallurgically sound. The manganese product is not a concentrate but an alloy, made directly from a lean unconcentrated ore. The customary method is to manufacture alloys from high-grade ores or from concentrates of leaner ores.

A discussion of the costs of ferro-grade manganese ores is presented in chapter 7. This discussion is in general terms only, but should be of use in so far as it relates to competitive conditions.

The results of a study of blast-furnace practice on ferromanganese and on spiegeleisen are presented in chapter 8. On account of the difficulty of obtaining complete and reliable records, the work is to be regarded as largely preliminary. Some of the conclusions tentatively advanced may be erroneous because of the nature of the data on which they are based; in the main, however, they are believed to be correct and are put forward in the hope that they may arouse criticism and thereby stimulate discussion and research. The Bureau of Mines proposes to continue this investigation.

The importance of allocating good grades of coke to manganese-alloy furnaces is discussed in chapter 9. The need for such allocations has passed but it is felt that the paper has more than an historic interest. Conservation and economy would both be served if the principles laid down were more effectively observed.

The chapters already cited deal with the manufacture of manganese alloys in the blast furnace with coke fuel. During the war there was considerable development in the electric smelting of such alloys. The Bureau of Mines was actively interested in the applicability of electric smelting to the leaner, more siliceous domestic ores, and conducted experiments at the Ithaca (Cornell University) field station. The results are presented in chapter 10. The general conclusions are that lean ores and manganiferous slags probably can not be smelted at a profit in the electric furnace except in times of high prices.

The purpose of the investigation described in chapter 11, on the use of manganese alloys in open-hearth steel practice, was chiefly to point the way to conservation of resources. The results show that conservation could best be attained by developing the use of those alloys, such as spiegeleisen and silicomanganese, which could be produced from lean domestic ores, thus conserving the high-grade domestic ores and reducing the need for high-grade foreign ores.

In addition to those members of the staff whose names appear on the several papers, acknowledgment is due to F. H. Probert, G. D. Louderback, Theodore Simons, W. S. Palmer, W. R. Eaton, F. B. Foley, C. F. Julihn, and all others who contributed to the success of the work by their active and whole-hearted cooperation.

CHAPTER 1.—GENERAL INFORMATION REGARDING MANGANESE.

By C. M. Weld.

USES OF MANGANESE.

Approximately 95 per cent of the manganese consumed in this country is used in making steel not only to deoxidize and recarburize the molten metal, thereby making possible the production of cleaner and sounder ingots containing the desired amount of carbon, but also to impart certain qualities to the finished product. Small proportions of manganese, 0.5 to 0.8 per cent, make the steel easier to work and stronger in service. At the same time slight proportions of impurities remaining in the steel are taken into combination and rendered less harmful. Relatively small quantities of so-called "manganese" steel are made which contain 11 to 14 per cent or more of manganese and possess special qualities of hardness and strength.

To be used in manufacturing steel, the ore must first be smelted to an alloy, in which the manganese is combined in varying proportions with iron, carbon, and silicon. Formerly the two standard alloys were ferromanganese, containing 78 to 82 per cent metallic manganese, and spiegeleisen, containing 18 to 22 per cent metallic manganese. During the war the composition of these two alloys underwent considerable modification, the standard manganese content being lowered in order to make practicable the use of leaner domestic ores. At the same time two other alloys, silico-manganese and silico-spiegel, assumed more or less prominence. The approximate range of composition of these four alloys is as follows:

Table 1.—*Range of composition of manganese alloys.*[1]

Alloy.	Mn.	Fe.	Si.	C.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Ferromanganese	50-80	40-8	0.5-1.5	5-7
Spiegeleisen	10-35	85-60	About 1.0	4-5
Silico-manganese	55-70	20-5	About 25.0	.35
Silico-spiegel	20-50	67-43	4-10	1.5-3.5

[1] Data largely from Newton, Edmund, manganiferous iron ores of the Cayuga district, Minnesota: Univ. of Minnesota Bull. 5, 1918, 126 pp.

The phosphorus must be so low that adding the alloy will not cause the phosphorus in the steel to exceed the specified limit, which is usually about 0.05 per cent. Consequently the phosphorus limit in the alloy will vary more or less directly with its manganese and carbon content.

Practice in utilizing the alloy varies widely. The alloy may be added to the molten metal either molten, red-hot, or cold, with or without other additions such as ferrosilicon or coal; it may be added in the open hearth, the converter, or the ladle. The average consumption of metallic manganese per ton of steel is approximately 15 pounds.

Minor uses of manganese ore are as follows: As an oxidizing agent in dry-cell batteries, for decoloration of glass, as a drier in certain paints and varnishes, in the preparation of oxygen on a small scale, and in the manufacture of certain disinfectants. It is also used to color glass, pottery, brick, and tile, and in calico printing and dyeing. Of these minor uses the first two named are by far the most important.

SPECIFICATIONS.

Ores suitable for the manufacture of alloys are generally termed "metallurgical" or "furnace" ores. They may have a wide range in composition, according to the particular alloy to be made from them.

Formerly, when the standard grade of ferromanganese contained 78 to 82 per cent metallic manganese, ores of ferro grade were required to contain not less than 40 per cent metallic manganese, dried at 212° F. Prices were based on not more than 8 per cent silica and not more than 0.2 per cent phosphorus, and ores containing more than 12 per cent silica or more than 0.225 per cent phosphorus were subject to rejection. When, in May, 1918, the standard content of ferromanganese was lowered to 70 per cent, the specified minimum manganese content of ferro-grade ores was correspondingly lowered to 35 per cent metallic manganese. At the same time the rejection limits for silica and phosphorus were considerably extended, but much more severe penalties were imposed for silica exceeding 8 per cent. The rejection of ores with more than 0.25 per cent phosphorus was left to the option of the buyer, who at the same time was urged to use the ore, whatever the phosphorus content, if he could use it to advantage. The effect of silica and phosphorus, and their relation to the alloy to be produced are discussed in chapter 3. The purpose of relaxing specifications was to widen the market for the leaner and more siliceous domestic ores. It is probable that former specifications will be revived with return of normal conditions.

Once it was the custom to pay for the iron in manganese ores of ferro grade, but this practice was discontinued several years ago. The permissible high limit for iron is governed by the nature of the alloy to be produced, as practically all the iron goes into the alloy and degrades it by crowding out manganese. In general, the allowable ratio of iron to manganese in the ore would be approximately 1 to 10 for 80 per cent ferro, or 1 to 5 for 70 per cent ferro.

Ores suitable for the manufacture of spiegeleisen may contain a much larger proportion of iron to manganese and are usually termed manganiferous iron ores. No exact specifications for ores of this class have been customary. In general, the silica and phosphorus requirements have been about the same as for ferro-grade ores, the balance consisting of manganese plus iron in varying ratio, plus gangue materials, such as alumina, lime, and magnesia. Ores containing 15 to 40 per cent metallic manganese were formerly classed as manganiferous, and these limits will probably be adopted again in the future. The war-time classification included ores containing 10 to 35 per cent manganese.

Certain manganiferous ores from the weathered parts of silver-lead deposits contain enough silver to warrant their shipment to lead smelters for use in fluxing. Hence these ores are not available to the steel industry.

When the alloy to be produced is silico-manganese or silico-spiegel a much higher content of silica is acceptable than with ores to be used in making ferromanganese or spiegeleisen, but the former alloys can advantageously be made only in the electric furnace. Ores with 30 to 40 per cent of manganese and 20 to 25 per cent of silica can be used to advantage in making silico-manganese. The ratio of silica to manganese may be still higher if the usual slag-making constituents are relatively absent. It is also probable that at least a part of the phosphorus in the ore may be volatilized in the electric furnace, thus raising the permissible limit of this element. Roughly, the same holds true for silico-spiegel, with the substitution of iron for a part of the manganese.

A large class of manganiferous ores comprises those that are essentially iron ores containing small proportions of manganese, generally about 5 per cent but occasionally as much as 10 per cent. These ores are not available for making manganese alloys, but enter into the manufacture of manganiferous pig iron, which in turn contributes its manganese to the steel made from it.

The ores suitable for oxidizers and chemical use are commonly called "battery," "chemical," or "dioxide" ores. As their function is to act as an agent for carrying oxygen, it is essential that they contain a large proportion of manganese dioxide, either as pyrolusite or other minerals that readily liberate oxygen. The following table shows the proportions of MnO_2 and available oxygen in the common manganese minerals.[a]

TABLE 2.—*Manganese dioxide and available oxygen in the common manganese minerals.*

Mineral.	MnO_2.	Available O.
	Per cent.	*Per cent.*
Pyrolusite	100.00	18.39
Psilomelane	42.46 to 77.33	7.81 to 13.06
Manganite	49.44	9.09
Braunite	43.11	7.93
Hausmannite	37.99	6.99
Rhodochrosite	None.	None.

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[a] Fermor, L. L., Manganese deposit of India: Geol. Survey of India, Memoirs, vol. 37, p. 598.

The two most important industrial uses of manganese ore as an oxidizer are in the manufacture of dry cells and of glass. Specifications generally call for 80 to 90 per cent MnO_2, but it is understood that ores with as little as 70 per cent MnO_2 were accepted during the war. Formerly it was customary to require that the iron content should not exceed 1 per cent, but here also specifications were greatly relaxed, particularly as it has been shown that the presence of several per cent of iron does not greatly affect the efficiency of the battery. Copper, nickel, and cobalt, on the other hand, are probably harmful when present in excess of a few tenths of 1 per cent, though there is some difference of opinion as to this. The point is discussed in the chapter following.

Leaner argillaceous and siliceous ores, with less than 40 per cent metallic manganese, are used for coloring pottery, tiles, and brick. The amounts used annually for this purpose and for paints, dyeing and printing calicoes, and other purposes are unimportant.

PRICES.

Prices paid for imported metallurgical ores have always been subject to individual contracts, based on the chemical and physical characteristics of the particular ores. The terms of these contracts have, of course, never been made public, but their general trend has no doubt been reflected in the price schedules for high-grade domestic ores issued from time to time by the Carnegie Steel Co. The latter are summarized in the following table:

TABLE 3.—*Carnegie Steel Company's price schedules for domestic ores.*[a]

Year.	Prices in cents per unit for percentages of manganese ranging—				Cents per unit of iron.	Silica standard.	Phosphorus standard.
	From 40 to 43.	From 43 to 46.	From 46 to 49.	49 and more.			
						Per cent.	*Per cent.*
1892	27	28–29	29–30	31	10	8	0.10
1897	25	26–27	27	28	6	8	.10
1903	22	23–24	24	25	5	8	.10
1906	27	28–29	29	30	6	8	.10
1910	23	24–25	25	26	5	8	.20
1914	....	24	25	26	....	8	.20
1915	36	40	43	45	....	8	.20
1916–17	46	50	53	55	....	8	.20
	From 38 to 42.	From 42 to 46.	From 46 to 50.	50 and more.			
1918	90	100	110	120	....	8	.20

[a] Data largely from Mineral Resources U. S., various years, U. S. Geological Survey.

These prices are based on long tons for material delivered at the furnace; material dried at 212° F. silicia penalty, 15 cents per ton for each unit over 8 per cent; phosphorus penalty, up to 1910, 1 cent per unit of manganese for each 0.02 per cent phosphorus over 0.10 per cent; thereafter, 2 cents per unit of manganese for each 0.02 per cent phosphorus over 0.20 per cent.

In May, 1918, a new price schedule was adopted by the Ferro Alloys Committee of the American Iron and Steel Institute and was approved by the War Industries Board. It became effective on May 28. It is given below in full as announced, including penalties and terms of payment, although it is no longer in force and its present value is chiefly historical.

PRICE SCHEDULE OF MAY 28, 1918.

Schedule of domestic metallurgical manganese ore prices per unit of metallic manganese per ton of 2,240 pounds for manganese ore produced and shipped from all points in the United States west of South Chicago, Ill. This schedule does not include chemical ores as used for dry batteries, etc.

Following prices are on the basis of delivery f. o. b. cars South Chicago, and are on the basis of all-rail shipments. When shipped to other destination than Chicago, the freight rate per gross ton from shipping point to South Chicago, Ill., is to be deducted to give the price f. o. b. shipping point.

Prices for ore dried at 212° F.

Content of metallic manganese, per cent.	Price per unit	Content of metallic manganese, per cent.	Price per unit.
35 to 35.99	$0.86	45 to 45.99	$1.12
36 to 36.99	.90	46 to 46.99	1.14
37 to 37.99	.94	47 to 47.99	1.16
38 to 38.99	.98	48 to 48.99	1.18
39 to 39.99	1.00	49 to 49.99	1.20
40 to 40.99	1.02	50 to 50.99	1.22
41 to 41.99	1.04	51 to 51.99	1.24
42 to 42.99	1.06	52 to 52.99	1.26
43 to 43.99	1.08	53 to 53.99	1.28
44 to 44.99	1.10	54 and over	1.30

For manganese ore produced in the United States and shipped from points in the United States east of South Chicago, 15 cents per unit of metallic manganese per ton shall be added to above unit prices.

Above prices are based on ore containing: Not more than 8.00 per cent silica; not more than 0.25 per cent phosphorus, and subject to:

SILICA PREMIUMS AND PENALTIES.

For each 1 per cent of silica under 8 per cent down to and including 5 per cent, premium at rate of 50 cents per ton. Below 5 per cent silica, premium at rate of $1 per ton for each 1 per cent.

For each 1 per cent in excess of 8 per cent and up to and including 15 per cent silica there shall be a penalty of 50 cents per ton; 15 per cent and up to and including 20 per cent silica, there shall be a penalty of 75 cents per ton.

For ore containing in excess of 20 per cent silica, a limited tonnage can be used; but for each 1 per cent of silica in excess of 20 per cent, and up to and including 25 per cent silica, there shall be a penalty of $1 per ton.

Ore containing over 25 per cent silica subject to acceptance or refusal at buyer's option, but if accepted shall be paid for at the above schedule with the penalty of $1 per ton for each extra unit of silica.

All premiums and penalties figured to fractions.

PHOSPHORUS PENALTY.

For each 0.01 per cent in excess of 0.25 per cent phosphorus there shall be a penalty against unit price paid for manganese of one-half cent per unit figured to fractions.

In view of existing conditions, and for the purpose of stimulating production of domestic manganese ores, there will be no penalty for phosphorus so long as the ore shipped can be used to advantage by the buyer. The buyer reserves the right to penalize excess phosphorus as above by giving 60 days' notice to the shipper.

The above prices to be net to the producer; any expenses, such as salary or commission to buyer's agent, to be paid to the buyer.

Settlements to be based on analysis of ore sample dried at 212° F. The percentage of moisture in ore samples as taken to be deducted from the weight.

PAYMENTS.

Eighty per cent of the estimated value of ore (less moisture and freight from shipping point) based on actual railroad scale weights to be payable against railroad bill of lading with attached certificates of sampling and analysis of an approved independent sampling chemist, balance on receipt of ore by buyer.

Actual railroad scale weights to govern in final settlement.

Cost of sampling and analysis to be equally divided between buyer and seller.

It will be observed that the value per long ton of natural (undried) manganese ore f. o. b. mine was to be calculated from this schedule in the following manner; (1) Multiply the percentage of manganese in material dried at 212° F. by the corresponding unit price as given in the table, adding 15 cents to this price if the ore was produced east of Chicago; (2) add premiums or deduct penalties, if any, for silica and phosphorus; (3) convert to wet or natural basis; (4) deduct freight per long ton to Chicago.

The following freight rates from points in the several Western manganese-producing States to Chicago were still in force in December, 1919. These were for carload lots with a minimum weight of 60,000 pounds.

TABLE 4.—*Freight rates on manganese ore from Western States to Chicago.* [a]

State.	Rate per ton of 2000 pounds.	Rate per ton of 2240 pounds.
Oregon	$11.00	$12.32
Washington	11.00	12.32
California	11.00	12.32
Montana	8.00	8.96
Arizona	9.00	10.08
Colorado	7.00	7.84
Nevada	10.00	11.20
Utah	9.00	10.08
New Mexico	7.00	7.84

[a] Personal communication from H. H. Porter of U. S. Shipping Board.

The ore prices given above applied only to ferro-grade ores. Prices for manganiferous ores have usually been subject to individual contract. In 1918 the better class of Leadville ores sold for about 30 cents per unit, with a premium of 6 cents per unit for iron and a moderate penalty for silica.

Comparison of the prices for ferro-grade ore with those for ferromanganese is of interest. Eighty per cent ferromanganese was worth $40 to $60 per ton in prewar times. The price from May to November, 1918, was $250 for 70 per cent material. In the tables below are shown (1) the price of ferro per ton; (2) the price per unit of metallic manganese in ferro; (3) the price per unit of 46 per cent manganese ore; (4) the unit price of metallic manganese in ferro divided by the unit price of metallic manganese in the ore, to show the ratio between the two. Since November, 1918, the market has been in a demoralized condition, quotations falling continuously and rapidly.

TABLE 5.—*Comparison of prices of alloy and ore.*[a]

Year. (1)	Price of 80 per cent ferro per ton. (2)	Price per unit of manganese. (3)	Price per unit of domestic ore containing 46 per cent manganese, delivered at furnace. (4)	Column 3 divided by column 4. (5)
1910	$40.49	$0.51	$0.25	2.04
1911	37.25	.44	.25	1.76
1912	50.40	.63	.25	2.52
1913	57.87	.72	.25	2.88
1914	55.80	.70	.25	2.80
1915	92.21	1.15	.40	2.88
1916	164.12	2.05	.50	4.10
1917	300.17	3.86	[b].75	5.15
1918	285.00	3.56	[b]1.23	2.93

[a] Data largely from Metal Statistics, 1918, published by the American Metal Market and Daily Iron and Steel Report.

[b] Estimated average.

As the last column in the Table 5 indicates, although the price of ore relative to alloy fell far below during 1916 and 1917, the price adjustments of 1918 brought this ratio back again nearly to the figures for 1913–1915; a fact that appears to controvert the opinion prevalent among producers that the proportionate prices of 1918 strongly favored the furnaces. In fact, the ratio shown for 1918 was possibly less favorable to the furnaces than that for 1913–1915, owing to the great increase of conversion cost through high costs of labor, fuel, and supplies.

The prices of "battery" ore also rose enormously during the war. Such ores were formerly worth $20 to $35 per ton. The best grades at last accounts were being sold for $80 to $110 per ton delivered.

STATISTICS.

In the following tables of imports, production, etc., the number of tons of contained metallic manganese are shown as well as the number of tons of ore and alloys, for in no other way can a true comparison be made. The metallic manganese in the ores is also multiplied by a factor representing the conversion loss from ore to alloy to give the "recoverable" manganese in a form suitable for use in steel manufacture. The recovery in making ferromanganese is assumed to be 73 per cent, and in making spiegel 65 per cent. Where the exact analysis of the ores and alloys is not known, the figures for tons of contained metallic manganese are estimated on the basis of the best data available.

TABLE 6.—*Imports of manganese ores, 1903–1918.*

(Long tons.)

Year.	Russia.	India.	Brazil.	Other countries.[a]	Total ores.	Total manganese.	Recoverable manganese, 73 per cent.
1903	5,576	35,900	76,910	....	146,056	55,350	40,450
1904	11,959	10,200	66,875	....	108,579	41,200	30,100
1905	24,650	101,030	114,670	....	257,683	114,340	83,500
1906	13,805	154,180	30,260	....	221,260	97,600	71,350
1907	1,000	95,300	52,922	....	209,021	71,950	52,500
1908	250	143,813	17,150	....	178,203	79,760	58,150
1909	14,486	145,140	35,600	....	212,765	95,800	70,000
1910	33,120	140,965	55,750	....	242,348	110,250	80,500
1911	19,103	106,580	41,600	....	176,852	81,590	59,600
1912	83,334	128,645	81,580	....	300,661	142,700	104,200
1913	124,337	141,587	70,200	8,976	345,090	168,142	123,000
1914	52,681	103,583	113,924	13,106	283,294	134,640	98,300
1915	....	36,450	268,786	8,749	313,985	142,680	104,000
1916	....	51,960	471,837	52,524	576,321	259,317	189,300
1917	....	48,975	512,517	68,450	629,942	282,500	206,400
1918	....	29,275	345,877	116,151	491,303	204,137	149,200

[a] Chiefly Cuba and Central America. In 1918 the sources were as follows (figures are long tons): Cuba, 82,974; Costa Rica, 9,680; Panama, 5,607; Mexico, 5,251; United Kingdom (probably reexported Indian), 4,362; Chile, 2,998; Argentine, 849; China, 3,000; Japan, 709; miscellaneous small lots, 721.

TABLE 7.—*Domestic production of manganese ores.*[a]

(Tons of 2,240 pounds.)

Item.	1912	1913	1914	1915	1916	1917	1918
Ferro grade: [b]							
Total ores	1,664	4,048	2,365	9,709	26,997	114,216	294,497
Total manganese	765	1,860	1,090	4,470	11,880	48,000	118,000
Recoverable manganese	560	1,360	795	3,260	8,670	35,000	86,000
Spiegel grade: [c]							
Total ores[d]	107,569	138,089	158,582	207,511	304,486	732,618	831,000
Total manganese	20,994	20,642	25,170	32,093	58,318	116,880	125,000
Recoverable manganese	13,650	13,420	16,360	20,850	37,900	76,000	81,200
Total recoverable manganese, both grades	14,210	14,780	17,155	24,110	46,570	111,000	167,200

[a] Data in part from Hewett, D. F., Our mineral supplies; manganese: U. S. Geol. Survey Bull. 666–C. 1917, p. 4, and in part from Mineral Resources U. S., U. S. Geol. Survey, for several years.

[b] Ferro grade, 1913–1917, 40+ per cent manganese; in 1918, grade was dropped to 35+ per cent manganese,

[c] Spiegel grade, 1913–1917, 15 to 40 per cent manganese; in 1918, grade was dropped to 10 to 35 per cent manganese.

[d] Includes manganiferous zinc residuum, but excludes manganiferous ores.

TABLE 8.—*Imports and production of manganese alloys, 1905-1918.*[a]

[Tons of 2,240 pounds.]

Year.	Ferromanganese.				Spiegeleisen.				Total manganese, both alloys.	Steel produced.
	Imports.	Production.	Total alloy.	Total manganese.	Imports.	Production.	Total alloy.	Total manganese.		
1905	52,841	62,186	115,027	92,020	55,457	227,797	283,254	56,650	138,670	20,023,947
1906	84,359	55,520	139,879	112,000	103,268	244,980	348,248	69,650	18,1650	23,398,136
1907	87,400	55,918	143,318	114,800	48,994	283,430	332,424	66,500	181,300	23,362,594
1908	44,624	40,642	85,266	68,150	4,579	111,376	115,955	23,200	91,350	14,023,247
1909	88,934	82,209	171,143	137,000	16,921	142,831	159,752	31,950	168,950	23,955,021
1910	114,228	71,376	185,604	148,300	25,383	153,055	178,438	35,700	184,000	26,094,919
1911	80,263	74,602	154,865	123,900	20,970	104,013	124,983	25,000	148,900	23,676,106
1912	99,137	125,378	224,515	179,700	1,015	102,561	103,576	20,700	200,400	31,251,303
1913	128,070	119,495	247,565	198,050	77	110,338	110,415	22,000	220,050	31,300,874
1914	82,997	106,083	189,080	151,250	2,870	79,935	82,805	16,560	167,810	23,513,030
1915	55,263	149,521	204,784	163,800	200	97,885	98,085	19,600	183,400	32,151,036
1916	90,928	221,532	312,460	249,970	....	194,002	194,002	38,800	288,770	42,773,680
1917	45,381	260,125	305,506	244,400	....	192,985	192,985	38,600	283,000	45,060,607
1918	26,200	333,027	357,227	274,000	....	283,853	283,853	51,000	325,000	42,212,000

[a] Data compiled from various sources, largely from the statistical bulletins of the American Iron and Steel Institute.

The figures in these tables tell briefly the history of the manganese industry for a number of years past. Thus, Table 6 shows how imports from Russia ceased early in the war and imports from India declined, whereas imports from Brazil increased enormously up to 1918, when restrictions on account of the shortage of ships caused a decrease. During 1918, however, imports from Cuba and from Central American countries practically doubled the 1917 figure.

The domestic industry shows the first effects of the war in 1916. From that time on, domestic production, both of ores and of ferromanganese, increased rapidly.

Table 8 shows that dividing the total pounds of metallic manganese in all alloys produced and imported during the 14 years, 1905 to 1918, by the total tons of steel produced during the same period gives a quotient of 15.5 pounds of manganese per ton of steel. As there were heavy stocks on hand at the end of 1918, it is probable that the true average figure for metallic manganese consumed per ton of steel is more nearly 15 pounds.

No comprehensive figures showing the consumption of "chemical" ore are available. The requirements of the United States are understood to be 25,000 to 40,000 tons annually. In the past these ores were largely imported from Russia. Recently they have come in part from domestic sources and in part from Japan, Brazil, and other countries.

CHAPTER 2.—USES OF MANGANESE OTHER THAN IN STEEL MAKING.

By W. C. PHALEN.

INTRODUCTION.

This report is divided into two parts—use of manganese dioxide ore; use of manganese bronze.

USES OF MANGANESE DIOXIDE ORE.

GENERAL REMARKS.

Estimates of the quantity of manganese ore used in the industries, other than metallurgical, range from 25,000 to 50,000 tons per annum. The latter figure is probably more nearly correct. Most of the ore used is the highest grade of pyrolusite (MnO_2) obtainable and commands a much higher price than metallurgical ore. The manganese dioxide ore is generally termed chemical manganese ore.

Manganese ore, as an oxidizing agent, is used in the manufacture of dry cells, as a decolorizer in certain kinds of glass, and as a drier in oils, paints, and varnishes. The ore is also used directly or indirectly in the manufacture of various manganese chemicals. Many of the textbooks describe its reaction with hydrochloric acid as a source of chlorine. It is no longer used commercially in making chlorine gas, as the latter is an abundant by-product in the manufacture of caustic soda and potash. Some chlorine may be made by this method when the object is the production, not of chlorine, but of manganese chloride. The ore has been mentioned as a soil stimulant, but its value in the fertilizer industry has not been established, and so far as known it is not a constituent of any commercial fertilizers. Only those uses of commercial importance are mentioned here.

THE DRY CELL.

GENERAL DESCRIPTION.

To show the importance of manganese dioxide in the dry cell, the general make-up of the cell, the chemistry of the processes involved, and the constituents entering into the reactions are described herewith.

The modern dry cell, commonly but erroneously called the dry battery, is a portable modification of the "Disque" Leclanche cell invented by Leclanche about 1868. That consisted of a porous cup containing a carbon electrode and a mixture of rather coarsely ground retort coke and pyrolusite. The cup was sealed, two vents being left for the escape of gases, and was placed, with a zinc rod, in a glass jar containing a solution of ammonium chloride. Because of the simplicity and ease of operation of this cell, many attempts were made to fix the electrolyte in various media, such as sawdust, gelatin, asbestos, and silicic acid, so as to make the cell portable.

In 1888 Gassner brought out the first really successful dry cell. The positive pole consisted of a cylindrical mass of ground pyrolusite and coke packed in a canvas bag around a carbon electrode. This was placed in a zinc container which also served as the negative pole, and a paste of plaster of Paris, zinc chloride, and ammonium chloride was poured under and around it. As soon as the paste had set the cell was sealed with a rosin or pitch composition.

The Gassner cell, though a great improvement over various other dry cells, did not meet with much favor because of its high internal resistance and low voltage. The highest current that could be obtained was about 6 amperes, and its voltage was about 1.3. Hence its use was limited to service requiring only small current drains.

DEVELOPMENT.

The advent of the gasoline engine greatly stimulated the development and production of the present type of dry cell. Ignition systems demanded a cheap portable cell able to recuperate after comparatively heavy current drains and by 1897 the manufacture of such cells had attained considerable volume. The present normal yearly requirements of dry-cell production include about 25,000 tons of high-grade manganese dioxide ore, an equal amount of carbon (petroleum coke and graphite) and 8,000 to 10,000 tons of sheet zinc, besides corresponding quantities of zinc chloride, ammonium chloride, paper, carbon electrodes, pitch, and sundry other substances.

METHOD OF MANUFACTURE.

In the modern American dry cell of the usual type, the negative pole is a cylindrical zinc can which also serves as a container. Several sizes of cells are on the market, the great majority, however, being about 2½ inches in diameter by 6 inches high (the so-called No. 6 cell). The inner surface of the zinc is lined with a special grade of absorbent paper, which acts as a reservoir for the electrolyte and as a diaphragm between the zinc and the positive pole. The depolarizing mass, called the "mix," is tightly tamped into the can, around a centrally placed carbon electrode, to about 1 inch from the top. This "mix" is composed of ground carbon (usually calcined petroleum coke and graphite) manganese dioxide ore, and the electrolyte. The mix with the carbon electrode, constitutes the positive pole. After the mix has been tamped, the paper is turned down over the mix; sand or sawdust is poured in, to a depth of about one-half inch, and the cell is sealed with a hot pitch composition. The object of the layer of sand is to provide an expansion chamber for the electrolyte and the excess gas, and also to provide a dry bed for the hot pitch.

CONSTITUENTS USED.

The sheets used for the cans are substantially pure zinc, prime western spelter being generally used, and are usually about 0.016 to 0.019 inch thick in the 2½ by 6 inch cells. Comparatively little of the zinc is used up during the service life of a cell. The mechanical strength to withstand filling and the fact that the can corrodes unevenly, in spots, patches, or streaks, must be taken into account when considering the necessary thickness. One of the problems of the dry cell manufacturer is to construct the cell so as to make this corrosion uniform.

Formerly, blotting paper was largely used for lining the cans, but at present most manufacturers use a special grade of pulp board. This is of ground wood pulp and sulphite pulp, and is 0.03 to 0.04 inch thick. The pulp board should be porous enough to allow the electrolyte to diffuse readily through it, but still retain the smallest particles of carbon and manganese; and it should be capable of absorbing several times its weight of water. Furthermore, it should obviously be free from metallic particles.

FUNCTIONS OF DIFFERENT CONSTITUENTS.

The depolarizing mass or "mix" is the vital part of the cell. To render the most efficient service its different components must be properly proportioned.

The ammonium chloride should be as pure as the usual chemically pure article. It should be substantially free from alkalies, sulphates, inert material, and heavy metals. When the cell is discharging, by virtue of the carbon, manganese dioxide, and zinc couple the ammonium chloride is split up into hydrogen, ammonia, and chlorine, which attacks the zinc can, forming zinc chloride. The reaction is: $Zn + 2NH_4Cl = ZnCl_2 + 2NH_3 + 2H$. The liberated gases cause polarization.

Artificial graphite is generally used, several grades of it being made for dry cells. The graphite does not enter into the chemistry of the cell, merely serving, with the coke, to render the mix more conductive. The coke used is calcined petroleum coke—that is, residues remaining in petroleum stills. When raw it is practically a nonconductor. However, on being calcined at a high temperature it becomes denser and as good a conductor as the better grades of amorphous natural graphite.

The relative fineness of the graphite, coke, and manganese ore and their distribution should be so uniform as to make the mix approach a solid porous mass. The current should flow along radial lines from the carbon electrode to every point on the surface of the mix adjacent to the paper lining. The ideal condition is to have each particle of manganese coated with enough carbon to render it a good conductor but still porous enough to permit efficient depolarization, while the voids should be filled with the porous coke.

The zinc chloride should be as free as possible from heavy metals. Its function is to depolarize the ammonia, which it does by the formation of double salts of zinc and ammonium chloride. No exact information is available as to the reactions involved, but it appears that a slightly soluble double chloride of zinc and ammonium is formed as the end product. During the earlier part of the service life of a cell, inefficient depolarization of the ammonia is probably often the cause of failure under heavy drains. A cell that has been short-circuited or subjected to a heavy drain smells strongly ammoniacal when opened. Another cause of dry-cell failure is the formation of a highly resistant, nonporous crust between the paper lining and the zinc can, probably through the formation of the double salt mentioned or the formation of zinc hydrate.

ROLE OF MANGANESE.

The manganese serves to depolarize the hydrogen. The reaction involved is usually given as follows: $2H + 2MnO_2 = Mn_2O_3 + H_2O$. The manganese reacts almost instantaneously in depolarizing the hydrogen, very likely while the latter is still nascent.

CHARACTER OF MANGANESE ORE USED.[a]

CHEMICAL REQUIREMENTS.

Several factors determine the suitability of manganese ore for dry cells. The ore should have a high available oxygen content present in the form of pyrolusite (MnO_2), should have a minimum amount of iron, and should be free from copper, nickel, cobalt, arsenic, and other metals electronegative to zinc. Copper is particularly harmful. If these impurities are present in the electrolyte or insoluble compounds they do no harm other than as inert or poor conducting materials. If soluble, however, their solutions diffuse to the zinc can, where they are deposited, forming an electrocouple which causes local and useless corrosion of the zinc and consequent deterioration of the cell. When the cell is in service, the deleterious action of these impurities is greatly hastened.

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[a] Storey, O. W., Determination of manganese dioxide in pyrolusite: C. F. Burgess Laboratories, Madison, Wis.

PHYSICAL REQUIREMENTS.

The physical properties of manganese ores influence their suitability for use in the dry cell. An ore should be somewhat porous to perform its function efficiently. A somewhat hard but porous ore is likely to give better results than a hard, dense ore, even though the latter is higher in available oxygen. In a dense ore the depolarizing reaction takes place only on the surface, whereas in a porous ore it can occur throughout the mass. Better service life is obtained from a dry cell containing rather coarsely ground ore, as this can hold more electrolyte than finely ground ore. Other factors of importance are the porosity of the ore and the fact that more contact resistance exists between particles of fine ore than those of coarse ore. Careful grading of the ore greatly influences the performance of a cell. The ore, therefore, should not be of an earthy nature like wad, as this mineral does not lend itself to efficient milling and grading.

Before the war the manganese ore used in dry cells was Caucasian pyrolusite. Common specifications called for material containing 80 to 85 per cent MnO_2 and less than 1 per cent iron. No particular attention was given to other ingredients, at least by most buyers, because of the purity and uniformity of the Caucasian ore. There are considerations other than the content of MnO_2 which determine the usefulness of manganese for depolarizer purposes, such as the screen analysis, hardness, density, and other physical qualities. Various manufacturers employ different specifications as to the screen analysis, a common specification being that the run of material shall pass through a 10-mesh or a 20-mesh screen. Some manufacturers specify the removal of the fine particles.

During the war manganese from many other sources was used in making dry cells because of the scarcity of the Caucasian ore. Most of these ores run lower in MnO_2 and higher in iron and have larger percentages of impurities, some of which are decidedly harmful. During the war users accepted material that ran 70 to 80 per cent MnO_2 and as high as 3 to 4 per cent iron. Users have found by experiment and manipulation how to get results with domestic ores and foreign ores other than Caucasian, closely approximating the results obtained with Caucasian.

An important source of manganese dioxide during the war was the ore from old dry cells, which was rejuvenated by processes generally kept secret by the firms employing them. Such processes doubtless contributed to the conservation of our high-grade domestic manganese ore.

In the manufacture of dry cells two classes may be recognized—the standard or so-called No. 6 cell, which is used for ignition, telephone, signal, and other similar purposes; and the small size, or flash-light type, which is used for portable lighting. The quantity of pyrolusite ores used in the standard No. 6 cell is far larger than that used in other sizes. In the manufacture of flash lights, which is growing rapidly, the higher grades of materials are required, such as 80 to 85 per cent ore that has been purified and also various grades of chemically prepared manganese dioxide and hydrates of manganese.

MANGANESE ORES IN THE CERAMIC INDUSTRIES.

GLASS MAKING.

Practically all the raw materials used in glass contain some iron, usually in the form of ferric oxide. The iron, when present even in small quantity, imparts to the glass a pale green color that increases rapidly in intensity as the iron content increases. If a colorless glass is desired, this green color must be removed by some decolorizer. Manganese, selenium, cobalt, and nickel are the most common decolorizers in use, and of these manganese has been most widely employed because it permits easy control of the color. In using selenium and nickel, the quantity must be carefully controlled, but these latter substances are desirable, especially in window and plate glass, because glass decolorized with manganese often changes to a pink color on exposure to the light. A decreasing quantity of manganese is being used by makers of tank glass, and its place is being taken by selenium.

The quantity of manganese used varies considerably, depending on the character of the glass, the method of its manufacture, the iron content of the raw materials, and the character of the manganese ore used. Each manufacturer has his own ideas on this subject. The quantity used is figured in terms of pounds of manganese dioxide per 1,000 pounds of sand, which constitutes 50 to 75 per cent by weight of the entire batch. The temperature employed in the glass-making process helps to determine the quantity of manganese dioxide used, for because of volatilization the higher the temperature, the more manganese is necessary. The maximum limit is 10 to 15 pounds of manganese dioxide per 1,000 pounds of sand, and the minimum may be 2 to 2½ pounds.

CHEMISTRY OF USE OF MANGANESE IN GLASS-MAKING PROCESS.

Compounds of manganese, when other coloring ingredients are absent, produce pink, purple, and violet hues according to the chemical nature of the glass. Manganese dioxide neutralizes the green color caused by iron compounds. Used in excess, it imparts an amethyst tint, and when used in considerable excess, the color is so dark as to appear black.

The neutralization of the iron tint by manganese dioxide is explained by some chemists on a physical, and by others on a purely chemical basis. The green tint is due to the presence of ferrous silicate. Some chemists think that this green compound is oxidized to the ferric silicate, which has an almost imperceptible pale straw-yellow color. According to this view, the oxidizing agent used must not completely decompose at high temperatures, and manganese dioxide seems to be the most available compound fulfilling this condition. At red heat, the dioxide loses one-third of its oxygen, leaving the tetraoxide (Mn_3O_4) which, at still higher temperatures, is an oxidizing agent.

Red lead and other oxidizing agents have not this decolorizing power. Hence some chemists have thought that the result is not due to oxidation, or chemical reaction, but is purely physical. It is possible, however, that other compounds may lose their oxygen at too low temperatures to be effective as oxidizing agents.

SPECIFICATIONS FOR MANGANESE ORE USED IN MAKING GLASS.

Before the war, the ordinary specifications for manganese ore used in glass making were 85 to 90 per cent manganese dioxide and less than 1 per cent metallic iron. Outside of these two ingredients, each manufacturer has his own requirements. Special glasses may require ore carrying more than 90 per cent manganese dioxide and less than 0.5 per cent iron. The higher the manganese content and the lower the iron, the better the ore is for glass making. In general, the grades of ore are similar to those used in making dry cells. Obviously, siliceous pyrolusite is not objectionable but carbonaceous pyrolusite is.

Manganese ore for glass making is sold in powdered, granulated, or lump form. There are objections to the lump form because of the time required to melt it into the batch. Powdered ore is used principally when the batch is melted in pots; lump, or granular ore is used when melting is done in tanks.

Before the war, high-grade pyrolusite for glass making and other chemical purposes was imported from Russia, Saxony, Japan, Nova Scotia, and other foreign countries. As the war progressed, such ore became scarce and, as a consequence, specifications were relaxed and low-grade ores were purchased. During the war some excellent domestic ore was developed, which found a ready market.

OTHER CERAMIC USES.

Another use for manganese ore in glass making has developed in the last few years, namely, for producing black glass used for ornamental purposes. About 3 per cent of ore is added to the batch in making this opaque glass.

Pyrolusite is added to the constituents of glazes and enamels to produce purple tints. Black enamels are those containing manganese. Manganese oxide is also used in brick making.

USE OF MANGANESE SALTS IN DRIERS.

DEFINITION.

Driers are substances, generally metallic oxides or their compounds, that are added to linseed or other drying oils at high or low temperatures to make them capable of readily absorbing oxygen from the air, or of drying by its action. Some chemists consider the action to be catalytic, the manganese compound acting as a catalyser or carrier of oxygen. The principal manganese compounds used as driers are: Manganese sesquioxide (Mn_2O_3), pyrolusite (MnO_2), also known in the trade as dioxide, binoxide or peroxide, manganese hydrate, sulphate, borate, resinate, linoleate, oxalate, and possibly other salts. Certain of the corresponding double salts of manganese and lead are often used.

Some persons claim that pyrolusite is now little used in driers because the manufactured hydrate, on account of its purity, gives better results. This claim does not agree with statements made by dealers in the trade. Of the various substances named above, each acts in a way peculiar to itself. These driers are added only in small quantities, usually less than 0.5 per cent.

MANGANESE DIOXIDE.

Manganese dioxide, extensively used as a drier, is marketed in two forms, the natural and the artificial. The natural mineral, pyrolusite, is simply ground to a powder with water and then dried. The mineral is essentially a peroxide, a class of substances containing more oxygen than is required to satisfy the valence of the metal present. This extra oxygen is loosely combined and readily enters into combination with oxidizable bodies. This feature in the composition of manganese compounds makes them useful in oil boiling, because the oxygen combines with the oil, oxidizing it, while some of the manganese dissolves and forms a compound with the linoleic acid of the oil. In consequence of this action manganese compounds are powerful driers. The quantity of manganese dioxide added in the process of boiling is small, not more than a quarter of a pound to a hundred weight of oil to get the best results. The use of the black dioxide, however, tends to make the oil dark.

MANGANESE SULPHATE.

The methods of preparing this compound and its uses are described on page 24. Rather less than one-half a pound is added to each hundred weight of oil or paint. Oil boilers use it largely as a dryer of pale boiled oils.

MANGANESE BORATE.

Manganese borate is perhaps the least objectionable of all the manganese salts used as drying agents, although the black oxides are more used. To make the borate, 1 part of manganese sulphate is dissolved in 10 parts of distilled water and a little of this solution is added to some soda solution to determine whether any iron is present. If pure, the manganese sulphate solution is added to the hot borax solution as long as a precipitate forms. The precipitate is filtered, washed with hot water, and dried.

MANGANESE RESINATE.

Resinates, formed by the combination of resin or rosin with certain metallic oxides, are much used in making varnish and paint and with the exception of the linoleates are most readily soluble in linseed oil. Manganese resinate is made as follows: Soda ash is dissolved in water and the solution is boiled with steam, the proper proportion of light coarsely powdered rosin is added, and then more soda ash. The clear solution is run into a clear manganese chloride, the resinate separating as a white flocculent precipitate, which is filtered, washed, and dried.

MANGANESE LINOLEATE.

Manganese linoleate is made by pouring a solution of a soap, made by boiling linseed oil and caustic soda into a solution of manganese chloride or manganese sulphate. The dark-brown plaster-like mass is liable to oxidation. When exposed to air, the surface becomes covered with a hard, rather insoluble, protective skin. The material should, therefore, be kept in tightly closed vessels. It acts both as a bleaching agent and a drier for linseed oil. One pound, mixed first with 5 pounds of linseed oil and the whole poured into 10 gallons of linseed oil at 250° F. gives a good drying oil.

The double resinates and linoleates of manganese and lead are also in use.

MANGANESE OXALATE.

Manganese oxalate is prepared by precipitating manganese salts with oxalate of soda or potash or by treating manganese hydrate or carbonate with oxalic acid. One of the advantages of the oxalate is said to be that it decomposes during oil boiling. The manganese dissolves in the oil in combination with linoleic acid, and the oxalic acid is decomposed with evolution of carbonic acid. One-fourth to one-half a pound may be used per hundred weight of oil.

USE OF MANGANESE IN MISCELLANEOUS CHEMICALS.

MANGANESE CHLORIDE.

Manganese chloride ($MnCl_2$) is prepared by the action of hydrochloric acid on the dioxide, chlorine being a by-product of the reaction. It is a rose red, deliquescent salt used in dyeing cotton cloth a manganese brown or bronze. The fabric to be dyed is soaked in the salt solution and passed through a caustic alkali, whereby manganese hydroxide is precipitated in the fabric and on subsequent oxidation turns brown. The material thus treated may be used also for subsequent dyeing by anilin black.

MANGANESE SULPHATE.

Manganese sulphate ($MnSO_4$) may be prepared on a large scale from the black dioxide by heating to redness with ferrous sulphate and subsequently extracting with water. The salt forms pink crystals which are readily soluble in water and are used in calico printing and in porcelain painting. It is also used as a drier for pale oils or for conversion into the oxalate or borate which are used for the same purpose.

MANGANESE PERSULPHATE.

Manganese persulphate, $Mn(SO_4)_2$ is prepared by the electrolytic oxidation of manganous sulphate ($MnSO_4$) and forms a black substance that can be obtained in solution only in the presence of sulphuric acid. It is used as an oxidizing agent in the manufacture of organic products.

POTASSIUM PERMANGANATE.

Potassium permanganate ($KMnO_4$) is prepared industrially by mixing a solution of caustic potash (KOH), specific gravity 1.44, with powdered manganese dioxide and an oxidizing agent, such as potassium chlorate. The mixture is boiled and evaporated, and the residue is fused in crucibles and heated until it has a pasty consistency. The potassium manganate (K_2MnO_4) thus obtained is dissolved by boiling with much water while a current of chlorine, carbon dioxide, or ozone is passed through the liquid. Potassium permanganate separates in crystalline form from concentrated solutions even in the presence of the caustic potash formed during the reaction, and is separated from the dissolved substances in a hydroextractor.

The permanganate is used for preserving wood; it is also used for bleaching textile fibers, by immersing them for a time in an aqueous solution of it and then dissolving the manganese dioxide with sodium disulphite. The permanganate is an energetic disinfecting and oxidizing agent and is used for purifying various gases.

USE OF MANGANESE IN MANGANESE BRONZE.

GENERAL REMARKS.

During the last twenty years manganese bronze has been widely used. The requirements of marine construction, of mining machinery, and wherever corrosion has presented a serious problem, have created a demand for a nonferrous metal to replace steel. Probably the most popular of such substitutes has been manganese bronze.

Manganese bronze made its first appearance about 1876. Its name is somewhat misleading for the alloy contains only a small percentage of manganese. Indeed, it is simply a brass to which have been added by proper methods of alloying, small quantities of aluminum, iron, or manganese, for the purpose of strengthening the alloy and making it denser and closer grained than the average yellow-brass casting.

FUNCTION OF MANGANESE IN MANGANESE BRONZE.

Manganese bronze should not contain much manganese, in fact not more than 0.05 per cent in high-grade bronze. Consequently the consumption of manganese in such alloys is small. The object of the manganese is not so much to act as an ingredient of the alloy, as to serve as a carrier of the iron necessary to insure the required strength and elastic limit. The manganese serves one purpose only, to introduce the iron, for without the manganese the iron would not alloy with the copper. Usually the manganese is added in the form of ferromanganese. If added in large quantities it hardens the alloy, but not nearly as much as iron; such an addition also lowers the elastic limit. Aluminum imparts a good sand-casting quality to the bronze.

Two grades of manganese bronze are now in common use. One is used for rolling into sheets, or drawing into wire or tubes, and for forging. This grade contains no aluminum, and has slightly less zinc than the other and can not be cast in sand. The second alloy is used for sand casting and is the one employed in making propellers and other common appliances.

The method of making manganese bronze—that is, the materials used, the methods of combining them, and the process for casting—is discussed by Sperry [a] and will not be discussed here.

McKinney [b] describes a process of manufacturing manganese bronze wherein is used, instead of virgin metals and raw materials of the highest purity, by-products and scrap. The methods described are, therefore, timely.

The composition of manganese bronze is as follows:

Composition of manganese bronze.

Constituent.	Per cent.
Copper	57.00–59.00
Zinc	38.00–40.00
Iron, manganese, aluminum, and tin	.25– 1.00
Lead	.10– .50

Evidently there is no particular need of using high-grade materials, provided the finished product is properly refined. Among the low-grade materials suggested for use in making manganese bronze are the following: Skimmings from the foundry, especially skimmings and dross ordinarily recovered from brass rolling mills or cartridge-case plants; zinc dross recovered from galvanizing plants; aluminum turnings that are generally unrecoverable without serious loss and deterioration of product through oxidation, and other by-products and scrap metals that ordinarily are not usable in foundry practice as remelting scrap.

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[a] Sperry, E. F., Manganese bronze and its manufacture: Brass World, vol. 1, December, 1905, pp. 399–406.

[b] McKinney, P. E., Manganese bronze: Am. Inst. Min. Eng. Bull. 146, February, 1919, pp. 421–425.

High-grade manganese bronze can not be made from the above raw materials on a small scale and thus manufacture in crucible furnaces is excluded. A reverberatory furnace or other equipment with which a bath of considerable proportion may be employed is necessary. McKinney [a] discusses a typical charge, the materials being melted in the presence of charcoal with salt as a flux.

USES OF MANGANESE BRONZE.

The most important use of manganese bronze is in propeller blades. A strong, tough alloy is necessary which will resist the action of sea water. The blades are made thin to save weight.

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[a] McKinney, P. E., Manganese Bronze: Am. Inst. Min. Eng. Bull. 146, February, 1919, pp. 421-425.

CHAPTER 3.—PROBLEMS INVOLVED IN THE CONCENTRATION AND UTILIZATION OF DOMESTIC LOW-GRADE MANGANESE ORE.

By Edmund Newton.

INTRODUCTORY STATEMENT.

In the past, the steel industry of the United States has depended almost wholly on imports for its supplies of manganese. Many of the important domestic sources yield ores that in their natural condition contain less manganese than the foreign ores the steel industry has been accustomed to use. To make these domestic ores available, therefore, they must be concentrated or practice in the steel industry must be modified.

Roughly, 25,000 tons of high-grade manganese ores is used annually for dry batteries, for chemical purposes, and in other minor ways, and approximately 750,000 tons is required for making steel.

By present practice every ton of steel takes an average of about 15 pounds of metallic manganese, which generally is added to the steel in the form of an alloy. The standard alloys are 80 per cent ferromanganese and 20 per cent spiegeleisen. During the year 1917, 286,000 tons of ferromanganese and 193,291 tons of spiegeleisen were made in this country, the former largely from imported ores; and 45,381 tons of ferromanganese was imported. The metallic manganese represented by these alloys was 304,000 tons, being roughly the product of 800,000 tons of high-grade ore and 345,000 tons of low-grade ore.

There is an abundance of low-grade ore in this country suitable for the manufacture of spiegel, but higher grade ore is necessary to make ferromanganese. For this reason the concentration of domestic ore presents a field for constructive and practical research.

MANGANESE DEPOSITS IN THE UNITED STATES.

Before the war manganese ore was mined in relatively small quantities in the Appalachian region, which includes parts of Virginia, Tennessee, and Georgia, and in Arkansas, but in consequence of higher ore prices because of the rise of ocean freight rates, manganese mining has been undertaken in Montana, California, Arizona, New Mexico, Nevada, Utah, and Minnesota as well as in the Appalachian region.

Data now available indicate that in this country deposits of high-grade manganese ores are usually small, but some deposits of ore lower in manganese and higher in iron are of considerable size. In the aggregate, the total quantity of manganese-bearing material is relatively large, but the difficulty of mining small deposits of the better grades of material and the seeming undesirability of low-manganese alloys in the steel industry, make the outlook for large production of manganese in this country uncertain.

As regards the geologic origin of the majority of the manganese and manganiferous iron ores in this country, Harder[a] states that they are largely the result of secondary concentration. Most of the ores of the Eastern United States, Arkansas, the Lake Superior region, Leadville and other silver districts, and of western California are of this type. The rhodonite and rhodochrosite in the unoxidized parts of the silver veins at Butte, however, are primary concentrations derived from igneous intrusion. The ores of northern Arkansas are largely reconcentrations from low-grade secondary deposits, derived by decomposition of crystalline rocks, and the California ores are concentrations within chert lenses of material originally present in a disseminated form.

Manganese-bearing materials of the United States may be roughly classified as follows:

1. Manganese ore proper.
2. Manganiferous iron ore.
3. Miscellaneous material:
    - (*a*) Manganiferous silver and lead ore.
    - (*b*) Zinc residuum from manganiferous zinc ore.

Manganese ore, as now defined by the trade, is material that contains more than 35 per cent manganese and is suitable for the manufacture of 70 per cent ferromanganese. Manganiferous iron ore contains less manganese and more iron. In general, the iron predominates, but there is no hard and fast line of demarcation between manganese ore and manganiferous iron ore. Manganese and iron are so closely associated in nature that all gradations from low-manganese, high-iron ore, to high-manganese, low-iron ore may be found in various deposits or in the same deposit.

Manganiferous silver ore is similar to manganiferous iron ore; it carries enough silver to make it valuable for that metal. Commercial considerations alone control the balance between the manganese or the silver value.

Zinc residuum is a by-product of the smelting of zinc ores from Franklin Furnace, N. J., which contain considerable manganese. After the zinc is removed the remaining product, called residuum, has nearly the same composition as natural manganiferous iron ore, and for years it has been smelted to spiegeleisen.

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[a] Harder, E. C., Maganese deposits of the United States: U. S. Geol. Survey Bull. 427, 1910, p. 4.

In general it may be said that the domestic manganese ores consist of an aggregate of minerals of manganese and iron with various impurities, such as silica, alumina, lime, and magnesia and accessory constituents, such as phosphorus, sulphur, silver, lead, and zinc. The determining characteristics of the manganese minerals and their relation to impurities are discussed in subsequent pages.

In the steel industry manganese is chiefly used in the form of alloys. A less important use is for increasing the manganese content of pig iron to give particular grades, as foundry irons, or to assist in metallurgical operations of certain steel-making processes, as in the basic open-hearth process. The alloys of manganese generally used in this country are ferromanganese, formerly containing 80 per cent, but now 70 per cent, metallic manganese, and spiegeleisen, having 15 to 20 per cent metallic manganese. The rest of these alloys consists principally of iron with small quantities of carbon, silicon, and phosphorus.

During the last few years ferromanganese has been gaining popularity among steel manufacturers, with spiegeleisen declining proportionately. Until recently approximately nine-tenths of the metallic manganese used in the steel industry was in the form of the standard 80 per cent alloy. Ferromanganese, or "ferro," as it is usually called, is easier to use in steel making than alloys containing smaller quantities of manganese, as the required quantity of that metal is contained in a smaller bulk.

The difficulty of obtaining ores suitable for the production of "ferro" has led to a consideration of the possibility of using what may be called intermediate alloys with manganese contents varying between 20 and 80 per cent. In the electric furnace certain alloys can be made with a relatively large content of silicon, in addition to the manganese and iron. The extent to which such alloys may satisfactorily be used in steel manufacture is not alone a technical or economic problem, but is largely controlled by the human element and the unwillingness of steel masters to deviate from long-established practice.

Phosphorus is undesirable in finished steel. In the manufacture of manganese alloys all the phosphorus contained in the ore will be recoverable in the alloy and will enter the steel when the alloy is used. It is permissible, however, for an alloy high in manganese to contain more phosphorus than one low in manganese, for less of the former alloy is needed to carry a given quantity of manganese. It is interesting to note that when 17 pounds of 80 per cent ferromanganese is added to a ton of steel the alloy may contain 1.33 per cent phosphorus and yet increase the phosphorus content of the steel by only 0.01 per cent. To add the same amount of manganese in the form of spiegel, the alloy may contain 0.33 per cent phosphorus and increase the phosphorus content of the steel 0.01 per cent.

For many years manganese alloys have been made chiefly in the blast furnace, although recently certain plants have produced them in the electric furnace. Blast-furnace practice on manganese alloys is generally similar to ordinary pig-iron practice, but there are high metal losses, principally in the slag, by volatilization and as flue dust. The amount of manganese contained in a given weight of slag may be partly controlled by furnace manipulation, but it is evident that the total amount of manganese lost in this manner is directly proportional to the "slag volume."

The results of increased slag volume and greater loss of manganese are cumulative and rather serious. More ore must be used per ton of alloy produced. This additional ore carries more slag-forming constituents. More coke is required to melt it, which in turn tends to produce more slag, and increased slag volume cuts down the daily output of alloy from the furnace. The greater loss of manganese decreases the ratio of manganese to iron in the alloy, so that unless proper allowance is made the alloy will be below the standard grade and therefore subject to a penalty by the purchaser. Not only will the alloy sell for less money, but the decreased daily output of the blast furnace will lessen the total profits.

A manufacturer of manganese alloys endeavors to protect himself against these decreased profits and adjusts his schedules and penalties for ore purchase with that end in view. Although he endeavors to equalize the effects of poor ores, the alloy producer would prefer to buy better ores at correspondingly higher prices.

CONCENTRATION OF DOMESTIC LOW-GRADE MANGANESE ORES.

The comprehensive term "concentration" as here used is intended to cover the improvement of low-grade ore by any suitable means preliminary to smelting. The requirements of metallurgical practice control the classification of manganese ore as low grade and high grade. Thus the term low grade may refer to a low content of manganese with respect to iron or to large quantities of nonmetallic impurities. The detrimental effects in metallurgical practice and the resulting penalties are the incentive for attempts to improve the ore or raise its grade before smelting.

FACTORS CONTROLLING THE POSSIBILITIES OF CONCENTRATION.

In order to interpret properly the possibility of concentrating at a rofit any type of manganese-bearing material, many technical and onomic factors must be considered. For a particular property, district, or class of material it is necessary to obtain data on these factors:

1. Character and size of the deposit,
2. Conditions affecting mining and marketing,
3. Character of ore material as affecting the possible improvement of grade,
4. Metallurgical value of crude ore and possible concentrate,
5. Commercial considerations.

SIZE AND CHARACTER OF DEPOSIT.

Obviously a large deposit of low-grade manganese ore would warrant considerable experimental work in order to determine satisfactory methods of treatment. Conversely, if a particular type of ore occurred in only one deposit of only a few thousand tons, the value to the industry of the product from such a deposit would be relatively small, even if concentration were feasible. Therefore, no undue proportion of time should be devoted to a concentrating problem that although of considerable technical and individual interest, gives little promise of furnishing any considerable part of the industry's needs.

The mineralized mass must be of such size and character as to justify the expenditure of money in its development and beneficiation and to promise return interest on the investment proportional to the risk taken. This factor of cost is of vital importance, and it is feared that under the stimulus of production incident to national needs during the war sound business principles have at times been temporarily forgotten.

CONDITIONS AFFECTING MINING AND MARKETING.

Not only must the quantity and character of material available be considered, but the natural factors controlling the mining methods, the transportation facilities, and the marketing of products must be carefully studied.

The manganese deposits of the United States, although widely scattered and comparatively small, may nevertheless be mined by relatively simple and therefore cheap mining methods. The mines are for the most part shallow, and extensive development and elaborate equipment are not necessary. Intricate problems of ventilation and drainage do not have to be solved, and if all operations are competently directed common mine labor will generally suffice. Small deposits mean short life, the ore is mined out rapidly and more or less crude mining practice prevails. The cost of mining will, however, be more or less governed by the necessity of selective mining, which in turn is determined by the variability of the ore, the feasibility of economic concentration, transportation facilities, and distance from a consuming center.

All these factors must be coordinated and their combined influence carefully studied before intensive production from individual properties begins. Concentration by eliminating waste may yield a product desired by the steel industry, but its cost may be prohibitive. Discarding waste may enable the producer to offset excessive freight rates, but geographic isolation will invariably handicap an enterprise.

Foreign ores will always find a market in the United States because they come from larger and more uniform deposits, are mined with cheaper labor, and the ocean freights are lower than rail.

CHARACTERISTICS OF ORE AFFECTING BENEFICIATION.

CHARACTER OF MANGANESE MINERALS.

Although many minerals contain manganese, only a few are commercially important. Usually it is rather difficult accurately to identify the manganese minerals in domestic oxidized ore. Several minerals may be intimately associated, and one may have been formed by alteration of another. The hardness of the individual minerals varies widely. Pyrolusite is soft and may be readily pulverized between the fingers. Hence difficulty might be expected in attempts to recover this mineral by the common processes of wet concentration. The other minerals are harder, but usually brittle. Although the character of the individual minerals is important, the association of the several manganese minerals with various gangue materials often has a more important bearing on methods of concentration.

IMPURITIES ASSOCIATED WITH MANGANESE MINERALS.

Wherever manganese ore is mined on a commercial scale, the product of the mine always contains impurities. Some of these are obvious on inspection; others may require chemical analysis for their determination. The impurities associated with manganese minerals may be classified as (1) those derived from associated rocks, or rocks partly replaced by manganese-bearing solutions, (2) those associated with the manganese in solution, and deposited simultaneously, and (3) those chemically combined with manganese in the mineral.

From the viewpoint of the metallurgist, all are impurities, and must be removed either before or by metallurgical treatment.

For convenience, the common impurities in manganese ores may be classified according to certain general physical and chemical principles, as follows:

1. Metallic. Iron, lead, zinc, silver, and in some ores, nickel, copper, and tungsten.
2. Gangue. "Basic" lime, magnesia, baryta, "acid" silica, and alumina.

3. Volatile. Water (atmospheric moisture and molecular water), carbon dioxide, and organic matter.

4. Miscellaneous. Phosphorus and sulphur.

The chemical behavior of these impurities affects metallurgical operations, and the physical form in which they occur controls the possibility of removal previous to smelting, and the choice of methods for accomplishing such removal.

The proportion of manganese to the useless or harmful constituents of an ore determines the value and desirability of the ore.

The presence of appreciable quantities of any impurity means that more ore must be mined and smelted in order to produce a given weight of manganese alloy. Some impurities are more detrimental to alloy manufacture than others.

Metallic impurities, of which iron is the most common, are usually reduced in smelting and are retained by the alloy. The quantity present naturally affects the character of the alloy produced, which in turn controls its desirability for use in steel manufacture. Other metallic impurities occur usually in such small amounts that they are not detrimental to the resulting alloy. Zinc is an exception. This metal is largely volatilized in smelting, and if it is present in appreciable quantities its fume condenses as oxide in the hot-blast stoves and may hinder furnace operation. Unless the furnace-top gases are washed, the stoves must frequently be cleaned, with consequent loss of time. When the price of zinc is high, the zinc oxide recovered from the stoves yields a substantial sum.

Silver, from the standpoint of the steel manufacturer, is neither detrimental nor advantageous to manganese alloys. The silver content of a manganese alloy has no value; consequently no credit is allowed the miner for silver contained in an ore to be used for manganese-alloy manufacture. In some ores the silver content is such that the ore has greater value for the lead smelter. The manganese then acts as a flux and the silver may be recovered by purification of the lead.

The gangue impurities classed as basic and acid may also be called "slag-forming impurities." In smelting, these impurities must be fluxed to form slag. Slag is usually considered a waste product, but it has important metallurgical functions, and just enough slag must be present for performing these functions properly and economically. An excess of slag must be avoided. In manganese-alloy manufacture the slag contains more or less manganese, the quantity of manganese thereby lost being dependent on the basicity of the slag, the temperature, and the slag volume. The first two factors control the quantity of manganese in a given weight of slag, and it is obvious that the greater quantity of slag will result in a greater loss of manganese. A large slag volume will rapidly decrease the daily alloy output of a blast furnace. The overhead charges must be distributed over a smaller daily tonnage of alloy and the profit from the sales for a given unit of time will decrease. It follows that more than the quantity of slag required to provide for the metallurgical functions is highly undesirable.

Silica is usually the predominant slag-forming constituent in domestic manganese ore. Some silica is reduced to the metallic state in smelting and is recovered in the alloy as silicon, but the larger part must be fluxed with lime, magnesia, or other bases to form slag. Manganese-alloy slags should be basic; hence a larger quantity of slag will be produced from an ore with acid gangue than in normal iron blast-furnace practice. Alumina is a slag-forming constituent and although usually classed with silica, it acts somewhat differently in the blast furnace. Brazilian ores are notably high in alumina, but most domestic ores contain relatively small quantities.

Lime, magnesia, and baryta in an ore are also slag-forming constituents, but they combine with the silica and alumina present and thereby reduce the quantities of those bases necessary in the form of limestone or dolomite for the furnace charge. Baryta is not common as a gangue mineral. It is not as strong a base as either lime or magnesia. While these constituents offset the metallurgical effects of silica or alumina, as regards evaluating an ore, they represent weight, and if the ore must be transported a considerable distance to the point where it is smelted, it is doubtful whether their value as bases would equal the additional freight charge. Limestone can generally be obtained at low cost close to the smelter.

Volatile impurities are removed from the top of a blast furnace largely by the surplus heat. It is desirable, however, in order to reduce the loss of manganese to keep the top of a manganese-alloy blast furnace cool. Volatile compounds are not particularly detrimental to smelting. When carbonate ores are being treated the case is somewhat different. Some metallurgists claim that in treating rhodochrosite ores the ratio of carbon monoxide to dioxide in the furnace gases is disturbed, which has a detrimental effect on the reduction of the oxides in the upper part of the furnace. It has also been suggested that the carbon dioxide driven off combines with carbon of the coke, forming carbon monoxide in the upper part of the furnace, and thus increases the coke consumption. Definite data are not available on these points.

In practice, all the phosphorus in the ore mixture and that contained in the coke and limestone is recovered in the resulting alloy. The permissible quantity of phosphorus in an alloy (the proportion that does not produce detrimental effects when the alloy is added to steel) has not been definitely determined. The higher the manganese content of an alloy, the larger the proportion of phosphorus that may safely be contained. Ordinarily steel makers desire as large a margin of safety as possible, and therefore have specified that the phosphorus in an ore shall not exceed a certain percentage.

Sulphur is usually present in small quantities in oxidized manganese ores, but in the primary rhodochrosite ores of Butte and other parts of the West there may be considerable quantities of sulphides of iron and zinc. Sulphur is not a serious factor, as the conditions of blast-furnace operation when making manganese alloys are such that sulphur combines with manganese or lime and is readily retained by the slag, only traces entering the alloy.

Knowing the effect of impurities in manganese ores on blast-furnace practice, the methods of eliminating them may now be considered. Ore dressing deals with the problems of separating deleterious or useless materials from the more valuable minerals, thereby raising the grade and reducing the quantity of the concentrated product. To accomplish this, it is essential that the physical and chemical characteristics of the ore be determined. These factors are conditioned largely by the type of deposit from which the ore is mined. As types of ore, entirely disregarding genesis, we recognize:

1. Rhodochrosite and rhodonite; carbonate and silicate ores, deposited in fissure veins or replacing original rocks.
2. Nodular ores, accretions of manganese oxide in soft plastic clays.
3. Manganese oxides deposited in small fissures or fracture planes, as braccia fillings, or as more or less impure beds.
4. Manganese oxides occurring as infiltrations, deposited in minute pore spaces, as partial replacements, or otherwise intimately mixed with the rock or gangue.

In the first class of deposits, the principal gangue impurity is silica, although sulphides of silver, lead, zinc, and iron are often found in appreciable quantities. The silica occurs both as quartz and chemically combined in rhodonite ores. In the carbonate ore, the carbon dioxide may be removed by calcination, thus effecting a concentration of manganese, reducing the bulk, and lowering the freight rate per unit of manganese contained in the original ore, but rhodochrosite decrepitates strongly when heated to a temperature where the oxide is formed, tending to produce an excessive amount of fines which is undesirable in practice. The breaking up of the particles by calcination will isolate some of the free silica which on account of the larger sized particles might be screened out. The sulphide minerals may occur in such quantity that it is desirable to remove them by gravity methods of separation.

In the second class of deposits, the nodules are of variable size and usually high in manganese. They do not appear to be contaminated internally with the inclosing material. The clays are soft, whereas the nodules are generally hard. This type is common in the Appalachian region of the United States. The clay may be separated from the manganese nodules by log washers, followed where necessary, and where the size of the deposits warrants the installation, by picking belts, crushers, screens, and jigs.

In deposits of the third class, the manganese minerals, although closely associated with the inclosing rock, are generally not contaminated by it and may be relatively pure. The method of treatment varies with the size of the manganese particles and the hardness of the rock, but does not differ essentially from the treatment of the second class. If there is little or no clay, the log washer will be omitted, and crushing, screening, jigging, and possibly tabling will make up the concentrating process.

If the manganese mineral is largely pyrolusite, and therefore friable and soft, crushing may produce an excessive proportion of fines difficult to recover by gravity or water methods of separation. If, however, the manganese mineral is hard, dense, and massive, and the inclosing rock is more friable, the problem is simpler. When the specific gravity of the minerals and the gangue approximate each other, wet concentration is difficult unless the particles differ decidedly in size.

Obviously the association of gangue materials with the desired mineral in ores of the fourth class is so intimate that the finest crushing imaginable will not permit separation by mechanical means. To this type the siliceous manganese ores of the Western States may be assigned. Ore-dressing tests have conclusively shown that where the silica is chemically combined with the manganese or where colloidal silica envelops the manganiferous particles, any wet process or gravity concentration will not give the desired results.

CONCENTRATION PROCESSES.

This paper does not describe in any detail actual ore-dressing or concentration practice. It is axiomatic that small deposits or mines of questionable life, as determined by tonnage and markets, do not warrant elaborate plants or the adoption of intricate beneficiation processes. A general classification of methods applicable to the manganese industry is given below. The processes mentioned are all preliminary to the greater and final concentration of the desirable elements in the blast furnace.

I. Simple methods of concentration.
  (*a*) Selective mining.
  (*b*) Hand picking.
  (*c*) Jigging.
  (*d*) Screening.
  (*e*) Log washing.
  (*f*) Water classification.
  (*g*) Roughing-table treatment.
  (*h*) Slime-table or vanner treatment.
  (*i*) Pneumatic separation.
  (*j*) Combination of two or more of the above methods.

II. Complex methods.
  (*a*) Magnetic separation.
    (1) Without preliminary thermal treatment.
    (2) With preliminary thermal treatment
  (*b*) Electrostatic separation.
  (*c*) Hydrometallurgical processes.
    (1) Leaching with various acids. Precipitation by chemical substances.
    (2) Leaching with various acids. Precipitation by electrolysis.
    (3) Leaching with various acids. Evaporation of solution and heat treatment in rotary kiln.
  (*d*) Preliminary thermal processes.
    (1) Drying, to remove hygroscopic moisture.
    (2) Calcining, to remove carbon dioxide or combined water.
    (3) Agglomerating fine concentrates, to make them desirable for blast-furnace use.
    (4) Volatilizing manganese at high temperatures in the presence of constituents that form readily volatile compounds.
    (5) Direct reduction of oxides by carbon, under temperature control.
  (*e*) Miscellaneous processes.
    (1) Flotation.
    (2) Use of heavy solutions.

There are many standard machines for the concentration of ores, but it is unwise to think that a certain machine will accomplish the necessary result on manganese-bearing materials. As the character of the manganese materials varies greatly in different districts, it is more logical to determine in detail physical, as well as the chemical, characteristics of the material. When such preliminary study has shown the nature of the impurity, and its relation to the manganese mineral, it is easier to outline a suitable method of treatment. The flow sheet, however, must be determined by experiment.

COMMERCIAL CONSIDERATIONS REGARDING BENEFICIATION.

If the technical possibilities of beneficiating any particular ore are favorable, it is then necessary to ascertain whether such an operation on a commercial scale would yield a reasonable profit. The cost of the plant and its installation must be justified either by the available ore in the deposit, or by the length of time during which the profit could be made. The amortization of capital and the interest on the investment must be included in the estimation of cost.

The effect of concentrating an ore is not always clearly appreciated. Concentration implies that an improvement of metallic content is made by the intentional elimination of impurities, but there is always a loss of the valuable mineral itself. When the grade of the product is increased, its weight is decreased. In other words, 2 to 25 tons of crude manganese-bearing material may be required to produce 1 ton of high-grade concentrate. The income results from the sale of the smaller quantity of concentrate, but chargeable against this will be the cost of mining some tons of crude ore, the cost of treating the ore, the freight to market, and the special overhead charges. Concentration, however, may be necessary to make the material marketable at all.

ESTIMATED COST DATA.

In order to illustrate the above points a little more clearly, the essential economic factors have been applied to three ores from California.

It is estimated that the three ores were obtained from desposits of such size that it would be possible to mine 75 tons of crude ore a day. In the table below the figures from actual concentrating tests have been combined with estimates of the cost of mining and treatment. The mining-cost figures are those reported by one of the three mines for the season of 1917.

A rough estimate of the cost of a small concentrating plant capable of treating 75 tons of crude ore a day is $20,000. It is assumed that this amount must be charged off in a year of 300 operating days. Instead of the amortization rate per ton being actually computed, the $20,000 and the interest charge of 7 per cent are distributed over the annual production of concentrates. The interest charge at 7 per cent is also distributed over the total tonnage of concentrates.

Royalty is considered to remain at $1.50 a ton of crude ore, even if the material is concentrated. Based on concentrates, it therefore increases proportionally to the ratio of concentration.

The value of crude ore and concentrates has been computed on the basis of the schedule approved by the War Industries Board in May, 1919. These prices no longer hold, and new computations should be made on the basis of existing prices. The methods to be followed are, however, indicated by the examples given. A freight rate of $11.20 per ton has been used from California points to Chicago on a long ton of 2,240 pounds.

All figures in these estimates are based on dry analyses and on a long ton of 2,240 pounds. Because of the length of haul, it is assumed that moisture would be negligible in both crude ores and concentrates. The figures follow in table 9.

TABLE 9.—*Estimated cost data applied to three California ores.*

MANGANESE ORE 1009.

	Fe	Mn	SiO$_2$	P
Analysis of crude ore, per cent	2.55	40.01	28.91	0.047

Concentration results:	Crude ore.	Concentrates.	Tailings.
Per cent by weight	100.00	50.55	49.45
Per cent manganese	38.59	51.10	25.81
Per cent manganese by weight	100.00	66.93	33.07
Ratio of concentration	1.987	1.00	

	Fe	Mn	SiO$_2$	P
Analysis of concentrate, per cent	1.48	51.55	13.14	0.053

Daily crude ore capacity, tons	75
Daily concentrate production, tons	38

Estimated cost of mining crude ore, per ton:	
Mining	$3.00
Tramming and loading	1.25
Royalty	1.50
Unloading	.25
Trucking	2.50
Overhead	1.75
Total	10.25

Value of crude ore, per ton:		
Manganese value, @ $1.02	40.01×$1.02=	$40.81
Silica penalty	7× 0.50=	3.50
	5× 0.75=	3.75
	8.91× 1.00=	8.91
Total		16.16
Net f. o. b. Chicago	$40.81− 16.16=	24.65
Net value, railroad shipping point, $11.20 freight	24.65− 11.20=	13.45
Net profit on sale of crude ore	13.45− 10.25=	3.20

Estimated cost of concentrating plant	$20,000
Estimated amortization period	1 year—300 days
Charge per ton concentrate	$20,000÷(300×38)=$1.75
Interest on investment at 7 per cent per ton of concentrate	(0.07×$20,000)÷11,400=0.12
Estimated cost of concentration per ton	$0.75 per ton of concentrates

Total estimated cost of concentrate on cars:			
Mining	@ $3.00 per ton of crude	1.987×$3.00=	$5.96
Tramming	@ .60 per ton of crude	1.987× .60=	1.19
Concentrating	@ .75 per ton of concentrates	1.000× .75=	.75
Amortization			1.75
Interest			.12
Royalty at $1.50 per ton of crude		1.987= 1.50×	2.98
Unloading at .25 per ton of concentrate		1.000= .25×	.25
Trucking at 2.50 per ton of concentrate		1.000× 2.50=	2.50
Overhead at 1.75 per ton crude		1.987× 1.75=	3.48
Total			18.98

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1009—Continued.

Value of concentrate per ton:	
Manganese value, at $1.24	51.55× $1.24=$63.92
Silica penalty	5.14× .50= 2.57
Net value, f. o. b. Chicago	$63.92− $2.57= 61.35
Net value, railroad shipping point, $11.20 freight	61.35− 11.20= 50.15
Net profit on concentrate per ton	50.15− 18.98= 31.17

Daily profit on crude ore	75×$3.20= $240.00
Daily profit on concentrate	38×31.17=1,184.46
Balance in favor of concentrating	= 944.46

Annual profit on crude ore, 300 days	300×$240.00=$72,000
Annual profit on concentrate, 300 days	300× 944.46×283,338
Balance in favor of concentration	=211,338

Annual production of crude ore, tons	22,500
Annual production of concentrate, tons	11,400

MANGANESE ORE 1011.

	Fe	Mn	SiO_2	P
Analysis of crude ore, per cent	1.27	32.01	39.55	0.029

Concentration results:	Crude ore.	Concentrate.	Tailings.
Per cent by weight	100.00	46.47	53.53
Per cent of manganese	32.19	45.05	21.02
Per cent of manganese by weight	100.00	65.05	34.95
Ratio of concentration	2.152	1.000	

	Fe	Mn	SiO_2	P
Analysis of concentrate, per cent	1.50	45.05	18.64	....

Daily crude ore capacity, tons	75
Daily concentrate production, tons	35

Estimated cost of mining crude ore, per ton:	
Mining	$3.00
Tramming and loading	1.25
Royalty	1.50
Unloading	.25
Trucking	2.50
Overhead	1.75
Total	10.25

Value of crude ore—Too siliceous.

Estimated cost of concentrating plant	$20,000
Estimated amortization period	1 year—300 days
Charge per ton of concentrate	$20,000÷(300×35)=$1.90
Interest on investment at 7 per cent per ton concentrate	(0.07 ×$20,000)÷10,500=.12
Estimated cost of concentration per ton	$0.75 per ton of concentrate.

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1011—Continued.

Total estimated cost of concentrate on cars:		
Mining, at $3 per ton of crude	2.152×$3.00=	$6.46
Tramming, at $0.60 per ton of crude	2.152× .60=	1.29
Concentrating, at $0.75 per ton of concentrate	1.000× .75=	.75
Amortization		1.90
Interest		.13
Royalty, at $1.50 per ton of crude	2.152× 1.50=	3.23
Unloading, at $0.25 per ton of concentrate	1.000× .25=	.25
Trucking, at $2.50 per ton of concentrate	1.000× 2.50=	2.50
Overhead, at $1.90 per ton of crude	2.152× 1.90=	4.09
Total		20.60

Value of concentrate per ton:		
Manganese value, at $1.12	45.05×$1.12=	$50.46
Silica penalty	7.0 × 0.50=	3.50
Net value	3.64× 0.75=	2.73
Total		6.23
Net value, f. o. b. Chicago	$50.46−$6.23=	44.23
Net value, railroad shipping point, $11.20 freight	44.23−11.20=	33.03
Net profit on concentrate per ton	33.03−20.60=	12.43

Daily profit on crude ore	
Daily profit on concentrate	35×$12.43=$435.05
Balance in favor	

Annual profit on crude ore, 300 days	
Annual profit on concentrate—300 days	300×$4.35.05=$130,515
Balance in favor	

Annual production of crude ore, tons	22,500
Annual production of concentrate, tons	10,500

MANGANESE ORE 1020.

	Fe	Mn	SiO₂	P
Analysis of crude ore, per cent	1.50	39.95	25.10	0.075

Concentration results:	Crude ore.	Concentrates.	Tailings.
Per cent by weight	100.00	33.14	66.86
Per cent of manganese	38.77	51.21	32.60
Per cent of manganese by weight	100.00	43.78	56.22
Ratio of concentration	3.018	1.00	

	Fe	Mn	SiO₂	P
Analysis of concentrate, per cent	1.23	51.55	11.65	0.074

Daily crude ore capacity, tons	75
Daily concentrate production, tons	25

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1020—Continued.

Estimated cost of mining crude ore, per ton:	
Mining	$3.00
Tramming and loading	1.25
Royalty	1.50
Unloading	.25
Trucking	2.50
Overhead	1.75
Total	10.25
Value of crude ore, per ton:	
Manganese value, at $1.00	$39.95×$1.00=$39.95
Silica penalty	7× .50= 3.50 5× .75= 3.75 5.1 × 1.00= 5.10
Total	12.35
Net f. o. b. Chicago	$39.95−$12.35= 27.60
Net value, railroad shipping point, $11.20 freight	27.60− 11.20= 16.40
Net profit on sale of crude ore	16.40− 10.25= 6.15
Estimated cost of concentrating plant	$20,000
Estimated amortization period	1 year—300 days
Charge per ton concentrate	$20,000÷(300×25)=$2.67
Interest on investment at 7 per cent per ton of concentrate	(0.07×20,000)÷7,500= 0.19
Estimated cost of concentration per ton	$0.75 per ton concentrates
Total estimated cost of concentrate on cars:	
Mining, at $3 per ton of crude	3.018×$3.00=$9.05
Tramming, at 60 cents per ton of crude	3.018× .60= 1.82
Concentrating, at 75 cents per ton of concentrates	1.000× .75= .75
Amortization	2.67
Interest	.19
Royalty, at $1.50 per ton of crude	3.018×$1.50=$4.53
Unloading, at 25 cents per ton of concentrate	1.000× .25= .25
Trucking, at $2.50 per ton of concentrate	1.000× 2.50= 2.50
Overhead, at $1.75 per ton of crude	3.018× 2.67= 8.06
Total	29.82
Value of concentrate per ton:	
Manganese value, $1.24	51.55× $1.24=$63.92
Silica penalty	3.65× .50= 1.83
Net value, f. o. b. Chicago	$63.92− 1.83= 62.09
Net value, railroad shipping point, $11.20 freight	62.09− 11.20= 50.89
Net profit on concentrate per ton	50.89− 29.82= 21.07
Daily profit on crude ore	75× $6.15=$461.25
Daily profit on concentrate	25× 21.07= 526.75
Balance in favor	Concentrating= 65.50

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1020—Continued.

Annual profit on crude ore, 300 days	300×$461.25=	$138,375
Annual profit on concentrate, 300 days	300× 526.75=	158,025
Balance in favor	Concentrating=	19,750

Annual production of crude ore, tons	22,500
Annual production of concentrate, tons	7,500

DISCUSSION OF ESTIMATED COST DATA.

ORE 1009.

The manganese content of this crude ore is within the range of the schedule. The silica content is a few points above, but it has been assumed that a small tonnage of such ore might be accepted. The value of the crude ore has, therefore, been computed.

By a concentration of approximately 2 to 1 a high-grade product can be obtained. Shipment of the crude ore shows a small profit per ton. On concentrate the profit per ton is comparatively large, and, although it weighs much less, the daily profit is shown to be much larger.

ORE 1011.

The manganese content of ore 1011 is below the range of the present schedule and the silica content is very high. It is, therefore, considered that this ore could not be sold as mined, but would require concentration. The concentrate is not very high in manganese, being 5.50 per cent lower than that obtained from ore 1009, although the ratio of concentration is nearly the same. The computation shows a daily profit by concentrating considerably less than for ore 1009.

ORE 1020.

The manganese content of ore 1020 is within the range of the present schedule and the silica content is only 0.1 per cent higher than the rejection limit. Therefore, the value of the crude ore has been computed. The manganese content of the concentrate produced is almost the same as for ore 1009, but the ratio of concentration is much larger. Hence, the profit per ton is less than for ore 1009, although more than for ore 1011. The daily profit is considerably decreased by the smaller output. Consequently, while there is a decided balance in favor of concentrating ore 1009, the balance for ore 1020 is comparatively rather small.

FINENESS OF CONCENTRATES.

The concentration tests of the three ores embodied crushing to pass 10 mesh, classifying, and treating on roughing tables. Ore buyers have said that they were not interested in such fine material.

Indeed, it would not be suitable as a major part of a manganese blast-furnace mixture, but up to 5, or possibly 10 per cent, it should not be excessively harmful. There would be a larger loss in flue dust, and if material of better structure were obtainable it would be greatly preferred by blast-furnace operators. Because of the larger losses attending the use of this fine material, if it were purchased at all, it would probably be penalized.

Possibly such fine concentrates could be agglomerated by briquetting, sintering, or nodulizing. But in practice, the application of any such process to a particular mine is rather doubtful. In the first place, a small daily output would not suffice to keep such a plant in continuous and effective operation. A sintering plant might be erected for the joint use of several mines, or might be erected independently and operated on a custom basis. The first cost of the equipment and the operating costs that would probably prevail in California being high, it is doubtful whether such plants would be profitable, unless there were some assurance of high prices prevailing for a considerable period of time.

At present nodulizing plants at iron blast furnaces in Chicago and elsewhere are running on flue dust. In an emergency they might be turned over to manganese concentrates, and the operating company reimbursed in some manner, possibly by a joint assessment of miner and alloy manufacturer. It is presumed that the sinter would be superior to natural manganese ores as mined and shipped direct, and that a certain part of the cost might be charged off against improved furnace practice.

The time required for the erection of a plant may be considerable, and private capital might not be attracted to an enterprise of this character unless there were some assurance that a given margin of profit might be possible for a period long enough to permit retirement of the original investment for the plant and the gaining of a satisfactory return.

CHAPTER 4.—PREPARATION OF MANGANESE ORE.

By W. R. Crane.

INTRODUCTORY STATEMENT.

The great demand for manganese during the war caused many persons to engage in mining and preparing manganese ores for market. As a result much money was spent on plants that were not adapted to the work. This condition was recognized by the Bureau of Mines early in 1918, and an effort was made to assist the operators of manganese properties by sending engineers into the field to advise as to the proper methods of mining and treating the ores.

A preliminary study of the operations showed that the methods in general use were wasteful; consequently, if practice could be improved it was estimated that the production of the various districts could be increased, by saving a large proportion of the mineral formerly lost—thousands of tons a year. With this object in mind, a standard washing plant was designed, which was based on the best and most successful practice in manganese properties in the country.

The methods employed in cleaning and concentrating manganese ores and the scale of operations depend largely upon the size and character of the deposits worked. The ores obtained from the different forms of deposits are so varied in character that were it economically possible to treat all of them, a wide variation in practice would be necessary. The irregular deposits, consisting largely of manganese and rock fragments inclosed in clay, are probably more largely treated than the regular forms as they produce the bulk of the ores treated. A relatively large amount of ore also comes from the regular or blanket formations, which consist of replacement deposits in the limestone bed rock.

The blanket deposits may be hard ore lying upon or within the limestone bed rock, or may be soft or wad ores. The deposits of wad, some of which are of large size, are usually of low grade and present special problems in washing and concentration; as yet no serious attempt has been made toward their economic treatment.

Washing and concentration methods are discussed herein in the following order: (1) Description of the practice in the various districts and (2) description of a standard plant the design of which is based upon the best practice in the treatment of manganese ore in this country.

The principal difficulty in treating the ore, or wash dirt, from the irregular deposits is the uncertain supply. Next in importance is the character of the manganese mineral and the proportion of ore to waste, such as clay, sand, and iron. The kind of ore best adapted to washing and concentration is the nodular or pebble-like forms of small size; next is the granular form; both are usually of convenient size for log washing and subsequent jig work. The kidney, dornick, and massive ores require reduction before treatment by log washers and on the picking belt, but owing to the size and purity of many of the masses, it is often possible to make a high recovery prior to treatment in logs and jigs.

On account of the uncertain and variable conditions mentioned, the tendency in treating manganese is to reduce the equipment to a minimum, both with regard to kind and number of parts employed. This is frequently done irrespective of the desirability of such limitation. Imperfect cleaning and excessive losses usually result from the curtailment of equipment. However, inadequate equiment of plants is not always due to the wish to limit the first cost, but often to lack of definite knowledge regarding proper methods of treating manganese ores, and the following of local practice, however poor it may be. It is, therefore, important to present details regarding a standard plant that incorporates the most useful and desirable features of the best practice in this country.

METHODS EMPLOYED.

The preparation of manganese ores may be divided into two separate and distinct parts—namely, dry mining, and washing or concentration.

DRY MINING.

Dry mining does not refer to mining, except indirectly, but more to the cleaning of the ore, or wash dirt, by screening it as mined. Not all wash dirt is adapted to dry mining, as some clays are wet and sticky, or occur in large masses, rendering their separation from the manganese mineral by screening practically impossible. Dirt that can be best treated by dry mining is dry and granular, and can be easily broken and separated.

The dry-mining method is unsatisfactory at best and is scarcely ever employed in operations of any size except as a temporary expedient prior to the erection of a washing plant. In small-scale operations where the deposits are small, or sufficient funds to properly equip the mine are lacking, such a method may be permissible. Also, deposits distant from a water supply may require dry mining if they are to be worked at all. In any event the method is wasteful, the loss of fine ore frequently amounting to 25 to 35 per cent of the recoverable mineral in the bank.

On the other hand, the high-grade manganese occurring in solution cavities can be mined and cleaned by hand with or without screening. In such deposits the bulk of the ore is of suitable size and is easily cleaned; however, even under the most favorable conditions, large losses result.

The proportion of manganese in the wash dirt is the controlling factor in dry mining; dirt having less than 10 per cent mineral can not be successfully separated by screening unless the bulk of the mineral is of large size.

WASHING AND CONCENTRATION.

The principal considerations affecting the washing and concentrating of manganese ores are: Quantity of wash dirt available, percentage of manganese in the dirt, and water supply. No plant should be erected until the available supply of ore has been determined within reasonably close limits; failure to test a property properly before erecting a plant shows lack of good business judgment. Ore may occasionally be mined and hand picked under exceptionally favorable conditions of occurrence even when the proportion of ore to clay is 1 to 50, or 2 per cent, but mining must be done on a large scale by machinery and the bulk of the ore must be in fairly large masses. Log washers and jigs can not readily treat materials that are leaner than 1 to 35; the ratio of ore to dirt should preferably be not less than 1 to 10, or 10 per cent, which is a fair average for wash dirt treated.

The design of a plant for the proper treatment of any manganese ore must be based upon the following points: Character of the manganese mineral and its physical properties and occurrence; the associated minerals, and the inclosing rock formations; and the practice previously followed, if any. Of only slightly less importance are the local conditions, such as lay of the ground, water supply, and timber for construction and other purposes.

The character of manganese ore varies widely in different parts of the same district, and even in openings on one property. The mineral may be dense and hard, or porous and hard, or porous and soft. The specific gravity also varies considerably, ranging from 3.5 to nearly 5; the average for the bulk of the mineral treated is between 3.5 and 4. The small rounded or nodular forms and the small granular pieces can be treated to advantage, but fine material and large masses require special treatment.

The principal minerals associated with manganese, as indicated in a previous chapter, are quartz and iron, mostly limonite; barite is of rare occurrence, although occasionally found. As the specific gravity of iron is similar to that of manganese it is practically impossible to effect separation mechanically; consequently it is com-

14. Revolving screen, for jig No. 2, ⅛-inch, 3/16-inch, and ¼-inch; oversize to cell No. 1, undersize to cells Nos. 2, 3, and 4, No. 2 jig.
15. Jig No. 1 (two cells); gate discharges to *12;* hutches to *12;* overflow to *5.*
16. Jig No. 2 (four cells); gate discharges to *12;* hutches to *12;* overflow to *5.*
17. Hand-picking table, where ore is sorted by attendants, who pick out iron and other impurities. The iron is sold as manganiferous iron ore.

PRACTICE IN THE BATESVILLE DISTRICT, ARKANSAS.

There were 11 washers built in the Batesville district, Arkansas, during 1917 and 1918. None of these did satisfactory work, largely because the builders and operators did not appreciate the necessity of providing screens for properly sizing the material treated on the jigs.

The practice in this district is simple and may be outlined as follows:

Flow sheet used in Batesville district, Arkansas.

Wash dirt to *1.*
1. Grizzly, size 5 by 6 feet, 10 to 14 railroad rails spaced 4 inches; oversize—waste rock thrown on dump, lump ore reduced to size that will pass through grizzly to *2;* undersize to *2.*
2. Double log washer, 28 feet long, slope 1 inch to 1 foot; discharge to *3;* overflow to *4.*
3. Revolving screen, 6 feet long by 30 inches in diameter, with ¾-inch perforations; oversize to *5;* undersize to *6.*
4. Waste bank.
5. Picking belt, 22 feet long by 24 inches wide; ore to *7;* waste to *4.*
6. Harz jig (four cells); size of cells 22 inches by 42 inches; gate discharges to *7;* hutches to *8.*
7. Finished ore bin.
8. Harz jig (two cells), size 28 inches by 36 inches; gate discharges to 7, hutches to 7, tails to *4.*

Rarely are more than two jigs employed in the plants for washing manganese ores, and in many plants only one jig is used. Revolving screens or trommels are commonly used, but flat screens are occasionally employed. In either case it is the exception rather than the rule to find more than one size of opening, which ranges from ¾ inch to 1¼ inches. The wide range in size fed to the jigs from such screens renders good separation next to impossible. Few of the jig products are clean, and they require extensive hand picking to complete the preparation. This hand sorting could be obviated by simply rerunning the hutch products of the rougher jigs; fairly clean products could then be made on the second or cleaner jigs. When only one jig is employed, few or none of the jig products is suitable for market. Fairly close sizing would greatly improve the work done by the jigs and reduce the expense of preparation by eliminating hand picking.

Probably a dozen hand jigs have been in use in the district at different times. As with the power jigs, incomplete separation of waste was the almost universal rule, little or no attention being paid to sizing of the feed.

PRACTICE AT A MILL AT PHILIPSBURG, MONT.

The mill of the Philipsburg Mining Co. is one of the largest in the West, having a capacity of 75 to 100 tons per day. The ores treated are the oxides mined at Philipsburg, Mont. The flow sheet of the mill follows:

Flow sheet of Philipsburg Mining Co. at Philipsburg, Mont.

Wash dirt (ore) to *1*.
1. Grizzly, 2-inch spaces; oversize to *2*, undersize to *3*.
2. Gyratory crusher, No. 4, 1¼ inches, to *3*.
3. Ore bin to *4*.
4. Revolving screen, 1¼-mm. mesh; oversize to *5*, undersize by elevator No. 1 to *6*.
5. Picking belt; clean ore to 7, rejected by elevator No. 1 to *6*.
6. Revolving screen, two sections—first, ⅜-inch mesh, second, ¾-inch mesh; oversize of first to second, undersize of first to *8*, oversize of second to *6*, undersize of second to *9*.
7. Clean-ore bin.
8. Revolving screen, 4-mm. mesh; oversize to *10;* undersize to *11*.
9. Bull jig; concentrates to *12;* overflow to *13*.
10. Middling jig; concentrates to *12;* overflow to *13*.
11. Revolving screen, 1½-mm. mesh; oversize to *14;* undersize to *15*.
12. Dorr classifier; spigot to *16;* overflow to *17*.
13. Rolls, ⅛-inch space, by elevator No. 2 to *18*.
14. Fine jig; concentrates to *12*, overflow to *13*.
15. Dewatering tank; spigot to *19*, overflow to *17*.
16. Frue vanner; concentrates by No. 3 elevator to *20*, overflow to *17*.
17. Settling pond.
18. Revolving screen; 2½-mm. mesh, oversize to two coarse jigs, undersize to *11*.
19. Distributor to *21*.
20. Drier to *22*.
21. Wilfley tables; concentrates to *16*, tailings to *23*.
22. Concentrates bin.
23. Waste bank.

DESCRIPTION OF A STANDARD WASHING PLANT.

The cleaning of manganese ores by washing and jigging can readily be accomplished and that, too, with slight change in the present practice. The flow sheet of a plant properly arranged to do satisfactory work is given below.

Flow sheet of standard washing plant.

Wash dirt to *1*.
1. Grizzly, 2-inch to 4-inch spaces between bars; oversize rock to *2*; manganese by sledge to *3*; undersize to *3*.
2. Rock dump.
3. Log washer, double log, 20 to 30 feet long; discharge to *4*; overflow to *5*.
4. Revolving screen, cylindrical or conical, perforations or mesh ½-inch and 3/16-inch; oversize, everything above ½-inch, to *6*; undersize, ½-inch to 3/16-inch, to *7*; 3/16-inch and smaller, to *8*.
5. Mud or settling pond.
6. Picking belt; rock to *2*; manganese to *9*.

7. Rougher jig, four cells; gate discharges to *9*; first hutch to *9*; second, third, and fourth hutches to *10*.
8. Sand jig, three cells; gate discharges to *9*; hutches to *9* or *10*.
9. Finished ore bin.
10. Cleaner jig, three cells; gate discharges to *9*; hutches to *9* or *11*.
11. Shaking table; finished product to *9*; tailing to *5*.

A standard washing plant having the above flow sheet may be called a single washer or unit, and has a capacity of 40 to 50 tons of finished ore per 10 hours. An increase in capacity of such a plant, aside from a small increase gained through crowding, would mean doubling, trebling, or quadrupling of most of the equipment listed, with corresponding increase in the volume of wash dirt treated.

By single washer is not meant a single log, but a single washer with double logs. Single logs do good work, yet in a plant as outlined, a double log should be employed.

The sizes of the various apparatus of a standard plant are indicated in the flow sheet; the sizes of screens and adjustments of the various apparatus for proper operation are given below.

LOGS.

The number of revolutions of logs varies from 12 to 15 per minute. The amount of water required varies widely with the nature of the bank dirt, but is usually 50 to 75 gallons per minute. The capacity likewise varies with the bank dirt; average dirt should yield 40 to 50 tons product per 10 hours. Roughly, 25 horsepower is required to drive logs.

SCREENS.

The number of revolutions is 15 to 20 per minute, being approximately the same for cylindrical and conical screens 36 to 48 inches in diameter. The power required for driving varies somewhat with feed and size of material handled, but is about one horsepower. The capacity of screens of the sizes given ranges from 45 to 55 tons per 10 hours for the smaller, and 50 to 75 tons for the larger. For the ordinary material treated the screen should be not less than 60 nor more than 72 inches long. If a larger number of sizes is desired, it might be better to shorten the length of the separate screening surface than to increase unduly the length of the whole screen.

PICKING BELTS.

The sizes of picking belts commonly employed, which have proved satisfactory for the usual range of work done, are: 30 feet long by 18 inches wide to 50 feet long by 30 inches wide. The speed of the picking belt ranges from 50 to 60 linear feet per minute. The capacity is limited by the character of the work; if the material sorted is largely ore and the waste rock is fairly coarse, the capacity is large, similarly where the percentage of ore is small and the individual pieces are large; but where the ore and waste are about equal in amount and there is a wide range in sizes, the capacity may be small. Occasionally two picking belts are used, one for coarse, the other for the finer materials, but such practice is neither necessary nor desirable in a standard plant.

JIGS.

The following data show the proper adjustment and operation of jigs:

Data regarding size and operation of jig equipment.

(Dimensions in inches.)

	Width.	Length.
Size of sieve compartments	20 to 24	30 to 38
Size of plunger compartments	18 to 22	30 to 38

	1st cell.	*2nd cell.*	*3rd cell.*	*4th cell.*
Size of sieve openings:				
Rougher jig	¼	¼	3/16	3/16
Sand jig	3/16	3/16	⅛	⅛
Cleaner jig	3/16	3/16	⅛	⅛
Length of stroke of jigs:				
Rougher jig	1½	1½	1¼	1¼
Sand jig	1	1	⅞	⅞
Cleaner jig	1	1	⅞	⅞
Number of strokes per minute:				
Rougher jig	150	150	150	150
Sand jig	200	200	200	200
Cleaner jig	200	200	200	200

Power for 4-cell jig, 2.5 to 3.0 hp.

Power for 3-cell jig, 2.0 to 2.5 hp.

Hydraulic water pipes: Main pipe 2 inches; branch pipes 1 inch.

Amount of water used by a jig, 350 to 400 gallons per minute.

Height of tailboard or dams, 3½ to 4½ inches.

Slope of bed (sieve) should not exceed 1 inch to length of bed; with heavy mineral, bed should be level. Slope should be against, not with movement of mineral.

Drop between tailboards or dams, 1 inch.

Height of gate discharge above bed, 2 to 2½ inches. This must be varied to suit conditions and character of material treated.

Jigs to have full or large capacity must be fed regularly; therefore jigs receiving the feed directly or indirectly from logs usually do unsatisfactory work for the reason that irregular feeding does not permit the maintenance of a bed of uniform depth. Further, the feeding of materials which vary from sandy to clayey in character also renders it difficult to keep a full bed; clayey ores tend to clog the jig beds and are preferably cleaned by trommels and jets of water.

With a suitably arranged plant, practically all materials treated may be handled by gravity, but in places elevators are required. Elevators, although adding to the equipment and the expense of operation, permit a better arrangement, in that more of the apparatus may be on the same level, and therefore more accessible.

Elevators are particularly useful for raising the hutch products from the rougher and sand jigs to the cleaner jigs. Should it be found desirable to employ a shaking table in reworking the hutch products from the cleaner jigs, elevators should also be employed in handling them.

TABLES.

It is doubtful whether tables can be used to advantage in the preparation of the usual run of manganese ore, as outlined and discussed. However, there is a well-defined field for their application in treating certain ores, such as the low-grade blanket deposits and breccia ores.

DRY CONCENTRATION.

Largely through the lack of an adequate supply of water a number of attempts have been made to clean manganese ores by dry methods. Probably the most extensive and elaborate equipment is that of the Southern Hill Manganese Co. on the Southern Hill property, in the Batesville district, Arkansas.

It is proposed to separate the ore from the clay and rock fragments by passing the material as excavated through a revolving drum, where it will be thoroughly dried and the clay pulverized. On being discharged from the drum the fine material will be removed by a revolving screen, which is a continuation of the drum. The coarse manganese and waste will then pass to a picking belt, where final separation is to be made. The smaller sizes of ore are to be treated on hand jigs. This plant has never been operated on a commercial scale; consequently no definite statement can be made regarding its practicability.

The water supply for manganese washing and concentrating plants is important, as many of the workable properties are often at considerable distance from an available supply. Pipe lines 3 to 4 inches in diameter are sufficiently large for a standard plant. The conservation of water by passing it through settling or impounding ponds usually solves the problem of water supply where the quantity of fresh water is limited. Once a plant has been put in operation the addition of 25 per cent of the total consumption not only provides for the loss due to wastage, but furnishes the required amount of fresh water for those steps of the process requiring clean water.

GENERAL SUMMARY OF CONDITIONS AFFECTING CONCENTRATION.

On account of the irregularity of manganese deposits and the uncertainty of an adequate supply of ore, great care should be exercised in connection with the various operations, particularly with respect to prospecting and mining. Similarly, preparatory to the erection of a washing or concentrating plant, the factors that have to do with the success of the work must be carefully considered.

The principal considerations affecting the cleaning of ores are:

1. The character and grade of ore.
2. The recoverable percentage of mineral.
3. Relative value of crude to cleaned ore.
4. Basis upon which royalty is paid, whether crude ore or the cleaned product.

Aside from the clay and other materials more or less intimately mixed with manganese considerable silica is associated with the ores. The silica may be "free" or "attached." The free silica can be readily removed by washing, but the attached silica, being embedded in the ore or attached to it can be separated, if at all, with difficulty.

High-grade ores, particularly when occurring in large masses, and soft ore, as pyrolusite, should receive the minimum preparation consistent with proper cleaning. Low-grade ores usually require much more careful treatment than the high-grade ores, and the work and expense of concentration depend largely upon the impurities present. Free silica is not difficult to separate from the manganese. Soft ore, or wad, although of high grade, is difficult to clean without great loss from fines, particularly when much fine sand is mixed with the clays and ore.

The recoverable percentage of mineral in the wash dirt depends largely upon the character of the mineral. Certain clays are readily broken and separated from the manganese, whereas others become pasty when washed, adhering tenaciously to the particles of mineral. As a rule, the larger the pieces of ore and the higher the grade the more readily is separation from the waste effected, owing probably to the smoother surfaces. Nodular ore of small and fairly uniform size is readily washed and jigged, but fragments from large masses and rough particles resulting from decay of limestone and possible partial solution of manganese are very difficult to clean satisfactorily.

The relative value of crude ore as compared with that of cleaned ore may be the deciding factor in determining whether a concentrating plant should be erected. During the past year considerable quantity of low-grade ores was shipped at a low price, simply because there was a market for it. The question is whether such ores could not have been raised in grade by concentration, so as to have brought a price that would have warranted the erection of a suitable plant. However, uncertainty as to extent of the deposits and the length of time the prevailing schedule of prices would be maintained did not foster experiments of this sort.

The grade of ore upon which royalties are assessed has been the cause of considerable trouble in different districts, but in most of the districts during the past year royalties were paid on all ores coming within the schedule unless otherwise specified. In the future high-grade ores will alone be subject to royalty charges.

In future careful mining in well-proved deposits of high-grade ore will be necessary in the various manganese districts. The ores mined will, in turn, require either close hand-picking or concentration in well-designed plants in order to produce a high-grade ore, low in silica and phosphorus. With a dependable output of such ore, it should be possible to continue operation in the face of foreign competition wherever freight rates to consuming furnaces are reasonably favorable.

CHAPTER 5.—LEACHING OF MANGANESE ORES WITH SULPHUR DIOXIDE.

By C. E. van Barneveld.

INTRODUCTORY STATEMENT.

Early in 1918 the Bureau of Mines, through its mining experiment station at Tucson, Ariz., began an investigation of possible methods of recovering manganese from the numerous deposits of low-grade, siliceous, secondary manganese ores that are scattered throughout Arizona, New Mexico, and Utah. The aggregate quantity of such ore is large, but as a rule these ores are not amenable to ordinary methods of concentration, and the proportion of ore that can be brought to shipping grade by sorting and screening is usually small. Much of this ore is found in localities tributary to the copper-smelting districts where waste sulphurous roaster gases are available. The possibility of using sulphur dioxide as a leaching agent for manganese had long been recognized, and it was thought that a process developed at the Tucson station for leaching certain copper ores with hot sulphurous fumes might be successfully applied to this type of manganiferous material. In May, 1918, some leaching tests were made at this station on manganese ores from the Patagonia district in Arizona. The results were sufficiently encouraging to lead to an extensive program of research into the possibility of developing an SO_2-leaching process which would produce a high-grade sinter containing over 60 per cent metallic manganese in the form of oxides, free from silica and phosphorus. The investigation had not been completed when the economic situation as regards manganese was entirely changed by the armistice and further experimentation was abandoned. However, sufficient work had been done to warrant certain conclusions. As these may have a scientific interest and a bearing upon other chemical and metallurgical operations, the following brief report is presented.

The work was undertaken with due recognition of certain facts:

1. That prompt action was essential.
2. That the process would probably have to be applied at smelting centers where sulphurous fumes were available as waste products, and that the cost of transportation of ores to the smelter would be a limiting factor in its application.

3. That a plant of 250-ton capacity would be the desirable unit and that in all probability two such units would be the maximum installation for any one smelter.
4. That from 4 to 6 months would be required for construction and that the cost of the plant would have to be written off over an operating period of 12 or even 6 months.
5. That the cooperation of the copper-smelting companies in Arizona and Utah and of the large consumers of manganese would be necessary in order to arrange for prompt plant construction; also that it might be possible and necessary to use public funds.

The investigation was carried through the laboratory stage on a sufficiently large scale to warrant as a wartime measure the construction on the results obtained of a commercial unit. On October 1 negotiations were under way for the erection of a 250-ton unit at one of the copper smelters, in cooperation with a large producer of lead-zinc-silver-iron-manganese ore in which only the fluxing value of the manganese was being realized. While this negotiation was in progress the rapid improvement in the military situation so changed the outlook that the undertaking was abandoned before final plans were drawn.

RESULTS OF TESTS.

The tests made included ores from the Clifton, Bisbee, and Patagonia districts in Arizona; also representative ores from New Mexico, Utah, Nevada, and California. The results of the tests are presented in Table 10 following:

TABLE 10.—*Results of tests of manganese ores by sulphur dioxide leaching with continuous-drum process.*

Source of ore.	Mn in heads.	Weight of ore taken for test.	Size of ore (mesh).	Pulp ratio.	Temperature of solution.	Rate of feed per hour.	Mn in tails.	Percentage of Mn recovered.	Remarks.
	Per cent.	*Pounds.*			° *C.*	*Pounds.*	*Per cent.*		
Parker, Ariz.	11.9	500	20	2-1	50	200	1.25	89.5	Highly siliceous ore containing 58 per cent SiO_2, 4 per cent Fe, 5 per cent CaO.
Bisbee, Ariz.	15.4	200	20	2-1	51	100	4.2	73	High lime ore (33 per cent CaO).
Do.	15.6	200	30	2-1	52	100	6.3	58	High iron ore (32 per cent Fe).
Do.	15.2	200	20	2-1	50	100	2.56	83	General ore.
Do.	7.42	100	20	4-1	47	100	1.85	75	Highly siliceous ore (59 per cent SiO_2).
Tombstone, Ariz.	16.6	102	20	4-1	47	50	.99	93.9	Silver-copper ore.
Do.	20.1	101	20	4-1	47	50	.85	95	Do.
McCarty, Delta, Utah	30.4	100	20	4-1	45	50	3.4	90	Contained 30 per cent SiO_2, 7 per cent Fe, 16 per cent CaO.
Hardshell, Patagonia	26.3	250	30	2-1	50	200	4.2	84	Silver ore (10 ounces Ag per ton).
Prince consolidated ore, Pioche, Nev.	11.5	100	20	2-1	50	100	.6	94.8	Lead-silver-gold ore containing 33 per cent Fe, 4 per cent CaO, 11 per cent SiO_2.
Wad ore, California	33.	....	20	4-1	50	....	2.3	93	Sample by C. E. van Barneveld.
Montgomery, Nev.	13.3	100	20	4-1	46	....	5.5	59	High lime ore (31 per cent CaO), containing considerable manganese as carbonate and silicate.

OBSERVATIONS ON RESULTS OF TESTS.

The following observations, based entirely on work done with dilute hot sulphurous gases (under 6 per cent SO_2) used countercurrent to the flow of the pulp, are presented:

THE ORE.

The general run of western ores may be described as impure psilomelane resulting from alteration of silicate or carbonate. The impurities may include numerous substances such as lime, silica (which may or may not be combined), and iron in the form of limonite or siderite.

Such an ore is readily leached with SO_2 and need rarely be crushed finer than 20 mesh to insure a good extraction.

THE PULP.

The pulp should be broken into a fine spray to insure contact between the mineral particles and the reagent. A leaching drum was designed at this station for the lixiviation of nonsulphide copper ores which gives the necessary agitation and contact.

A patent (U. S. patent No. 243015) was applied for and was granted on November 8, 1918.

Briefly, this apparatus comprises a drum set horizontally, to rotate slowly. The interior is divided into compartments by a series of transverse partitions, each compartment having longitudinal baffles and peripheral lifters. The transverse partitions are perforated so that the pulp may pass through the successive compartments to the end or discharge compartment. The pulp is fed into the first compartment and drops to the bottom of the compartment; as the drum rotates the peripheral lifts raise the pulp until it is spilled onto the horizontal staggered baffles below in such manner that the pulp is splashed and distributed over the surfaces of the bars and against the transverse partitions in fine descending drops and particles, thereby insuring intimate and prolonged contact between the ore particles and the countercurrent sulphurous and oxidizing gases. With a constant feed a flow is established within the drum, so that the pulp is gradually passed through the successive compartments into the last or discharging compartment, the lifters of which raise the pulp and drop it into a discharge or exit pipe whereby it is conducted through a trap opening into a discharge launder.

The pulp enters one end of the drum cold, and the SO_2-charged air or gas enters the opposite end hot. The countercurrent flow developed results in the pulp becoming progressively warmer and the gas becoming correspondingly cooler, until the pulp at the discharge end is heated to any desired temperature. When SO_2 is introduced as a hot dilute gas containing less than 6 per cent SO_2 by volume (the balance of the gas being largely air, which has lost part of its oxygen and which may contain the various impurities commonly found in roaster gases) and is projected into a fine spray of hot pulp or solution, absorption of SO_2 is practically negligible and the hot pulp or solution will contain practically no free SO_2 and will discharge from the drum in a practically neutral condition.

SOLUBILITY OF THE MANGANESE MINERALS.

The various higher manganese oxides, especially pyrolusite, psilomelane, and wad, were found to be readily soluble in the hot sulphurous acid formed by introducing hot air or furnace gas containing 2 to 6 per cent SO_2 (by volume) countercurrent to a pulp flow having a consistency of 2 of water to 1 of ore crushed to 20 mesh. Manganite and braunite are not commercially soluble.

Silicates of manganese are insoluble in sulphurous acid at atmospheric pressure and only slightly soluble under pressure.

Carbonates of manganese may, from an operating standpoint, be considered insoluble. The reagent slowly attacks and decomposes the carbonate, but some of the dissolved manganese at once reprecipitates as a sulphite; this precipitate coats the remaining undissolved carbonate and effectually prevents further dissolution.

IRON.

The effect of SO_2 on iron minerals differs markedly from its effect on manganese minerals. Iron in the oxide form, such as magnetite, hematite, and limonite, is practically insoluble in SO_2. Metallic iron and iron in the form of carbonate (siderite) is readily soluble. This iron will be in solution as a ferrous sulphite or sulphate; it will, however, precipitate as a basic ferric compound in the presence of either $CaCO_3$ and oxygen or $ZnCO_3$ and oxygen; it will also be similarly precipitated from solution so long as there is undissolved MnO_2 in the ore charge.

PHOSPHORUS.

In some forms phosphorous dissolves readily in SO_2. No large scale tests in the drum were made on high phosphorus ores. Laboratory tests, however, indicate that by the use of tandem drums phosphorus may be eliminated in the following manner: The phosphorus will not be dissolved in the first drum (the discharge from which still contains much undissolved manganese and other bases). Whatever dissolved phosphorus is discharged from the second drum will return to the ball mill. The addition of large quantities of fresh ore will at once neutralize these solutions, and the phosphorus will thereupon precipitate. If sufficient soluble iron and phosphorus are present to cause the building up of these elements by resolution in the drum after precipitation in the ball mill, then the filtrate from the second drum may be passed over any available carbonates, such as $CaCO_3$, prior to being returned to the ball mill. The iron and phosphorus in solution would then be precipitated and eliminated from the circuit.

LIME.

In the presence of an appreciable amount of MnO_2 the concentration of free SO_2 is insufficient to decompose $CaCO_3$. As dissolution of MnO_2 progresses, some $CaCO_3$ will be dissolved and precipitated as $CaSO_4$ (insoluble). The catalytic action of the iron and manganese oxides will convert a small proportion of the entering SO_2 to SO_3; the resulting sulphuric acid will convert $CaCO_3$ to $CaSO_4$. Some of the $CaCO_3$ particles become coated with gypsum and are not further acted upon by the acids in the drum.

ZINC AND COPPER.

Zinc and copper in nonsulphide form are readily dissolved by SO_2, but sulphides of these metals are not attacked.

TREATMENT OF THE PREGNANT SOLUTION.

The discharge from the drum will consist of a pulp containing in solution as sulphates the manganese, zinc, copper, and perhaps some iron; as solids the gangue, the sulphides, the other insoluble minerals, and any dissolved lime reprecipitated as $CaSO_4$.

The pregnant solution may be separated from the solids by filtration and washing in pressure filters.

Any copper present in the filtrate may be recovered by precipitation on iron.

The manganese-zinc content is removed together, either by crystallization, in large-scale permanent installations, where the lowest operating costs are to be sought, or by evaporation in pans with artificial heat in smaller or in temporary installations, where minimum installation expense would regulate the choice.

The resulting sulphate is roasted for two hours, more or less, in a rotary type of clinker furnace at a temperature ranging from 825° to 1,000° C.

The product from the roaster is a hard, compact clinker which in the zinc-free ores will run from 60 to 64 per cent manganese. Any remaining manganese sulphate may be removed by water leaching; this leaching does not seem to cause the cake to crumble. The waste heat from this operation is used to heat the cast-iron evaporation pans. The SO_2 gas may be used over again as a solvent if desired.

The manganese sulphate commences to decompose around 700° C. and should break up completely at 830° C. It was found that in order to effect complete decomposition of the sulphate with reasonable quickness (say in 30 minutes) the roast should be preformed at considerably higher temperature—around 950° C. Plenty of oxygen is necessary. In a reducing atmosphere some manganese sulphide will be formed; once formed, this can only be broken up by first reoxidizing to the sulphate.

CONDITIONS ESSENTIAL IN COMMERCIAL PRACTICE.

Three essential conditions must be complied with in commercial operations.

(*a*) In order that dissolution may be effected rapidly a reasonable strength of gas is required—not less than 2 per cent and up to 6 per cent of SO_2—the balance of the gas being heated furnace products.

(*b*) Heat aids dissolution. It is desirable to have an admission temperature of gas which will give the discharging solutions a temperature of 40° C. minimum to 60° C. maximum.

(*c*) The time of contact is regulated by the rate of flow of pulp through the drum. The efficiency of extraction is directly proportionate to the pulp surface exposed to the gas. This in turn depends upon proper agitation and breaking up of the pulp into a very fine spray which must be so directed countercurrent to the gas that no particle of mineral can escape contact with the reagent. This can be satisfactorily accomplished in 15 to 30 minutes in the drum described.

LEACHING IN TWO STAGES.

Manganese sulphate solutions are near the point of saturation when they contain 11 per cent manganese (specific gravity 1.5). As the solution approaches the point of saturation the rate of dissolution decreases markedly. On the other hand, maximum concentration of solutions minimizes the expense of subsequent evaporation to dryness, which is the next step in the process. Therefore, leaching should be performed in two stages in two tandem drums as indicated on the flow sheet shown in figure 1.

Wet crushing in ball mills is advised in circuit with drag classifiers, the filtrate from the second drum discharge being used to pulp the ore.

The discharge from the first drum will be a practically neutral pulp made up of partly leached ore, which will still contain considerable undissolved manganese, and a solution containing $MnSO_4$ near the point of saturation, and also $ZnSO_4$ and $CuSO_4$ if nonsulphide zinc and copper are present in the ore. This solution will not contain iron. This pulp is then filtered on a pressure type of filter and the various products are disposed of as follows:

(*a*) The filtrate is treated for recovery of valuable metals as previously described.

(*b*) The filter cake and wash water are repulped in a pulping tank, together with the wash water from the second drum filtration, to form the feed for the second drum, where the remaining SO_2-soluble minerals, including perhaps some iron, are dissolved.

The discharge from the second drum is filtered and the various products are disposed of as follows:

(*a*) The residue or filter cake goes to waste, or to a briquetting plant in case the lead-silver or copper-silver or other values warrant further treatment such as smelting.

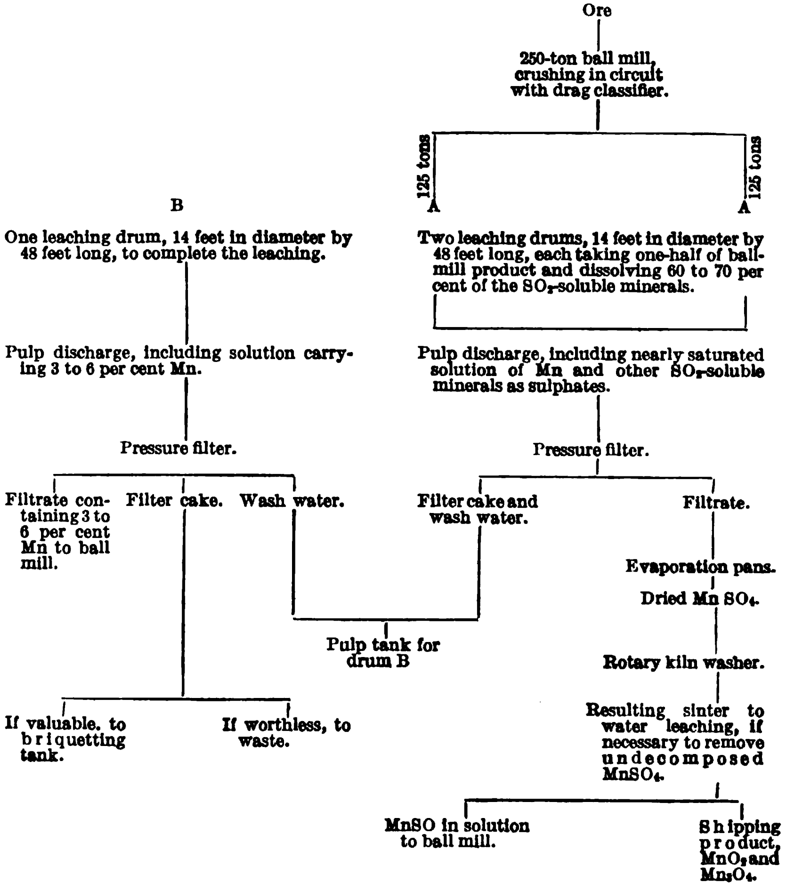
Ore
250-ton ball mill, crushing in circuit with drag classifier.
125 tons
125 tons
A
A
B
One leaching drum, 14 feet in diameter by 48 feet long, to complete the leaching.
Two leaching drums, 14 feet in diameter by 48 feet long, each taking one-half of ball-mill product and dissolving 60 to 70 per cent of the SO_2-soluble minerals.
Pulp discharge, including solution carrying 3 to 6 per cent Mn.
Pulp discharge, including nearly saturated solution of Mn and other SO_2-soluble minerals as sulphates.
Pressure filter.
Pressure filter.
Filtrate containing 3 to 6 per cent Mn to ball mill.
Filter cake.
Wash water.
Filter cake and wash water.
Filtrate.
Evaporation pans.
Dried $MnSO_4$.
Pulp tank for drum B
Rotary kiln washer.
If valuable. to briquetting tank.
If worthless, to waste.
Resulting sinter to water leaching, if necessary to remove undecomposed $MnSO_4$.
MnSO in solution to ball mill.
Shipping product, MnO_2 and Mn_3O_4.

FIGURE 1.—Flow sheet of proposed plant for SO_2 leaching of manganese ores.

(*b*) The filtrate, which may contain up to 5 or 6 per cent manganese, goes to the ball mill to pulp the original ore for the first drum.

(*c*) The washwater from filtration goes to the pulping tank in which the filtercake from the first filtration is pulped prior to being fed into the second drum.

The crushing, leaching, filtration, and washing are thus carried on in a circuit. Any iron dissolved in the second drum is returned to the ball mill and is there immediately precipitated.

FILTERS.

A pulp made of 20-mesh ore and a saturated, slimy $MnSO_4$ solution presents an interesting filtration problem. Tests on a semi-commercial scale proved that the vacuum type of filter could be used only by the addition of filter cells and by dilution of the $MnSO_4$ solution considerably below the saturation point. In a series of tests made under the direction of A. W. Hudson, of the Phelps Dodge Corporation, on manganiferous silver ore from Tombstone, Ariz., vacuum filters proved inadequate. The addition of filter cells partly nullified the concentration of silver values obtained by extraction of the manganese. The dilution of the pregnant liquid appreciably below the point of saturation increased the cost of subsequent evaporation of $MnSO_4$.

A test made on a pressure type of filter gave the following results: The quantity of pulp passed through the filter contained 3,200 pounds of solid matter and yielded 4,000 pounds of cake containing 20 per cent water. The filter yielded 6.6 pounds of filter cake (20 per cent water) per square foot per hour. On this basis a plant to treat 250 tons of dry ore would yield 312.5 tons of filter cake (20 per cent water) and would require $(312.5 \times 2{,}000) \div (6.6 \times 24) = 3{,}946$ square feet of filtering surface.

CORROSION.

Corrosion of metal in contact with the solution was found by experiment to be due entirely to the action of ferrous sulphate and liberated sulphuric acid. Leaching in two stages presents a simple solution of this difficulty. As sulphur dioxide has a marked affinity for manganese, conditions in the first drum may be so regulated as to avoid the dissolution of any iron, and the resulting pregnant liquid will not corrode the evaporation pans. The solution from the second drum may be neutralized and the dissolved iron may be precipitated over carbonates prior to the return of this solution into the crushing end of the circuit, or this may be done in the ball mill.

SULPHUR DIOXIDE.

Sulphur dioxide for leaching purposes may be obtained (*a*) by burning flower sulphur in sulphur burners; (*b*) by roasting high-grade iron pyrites containing sufficient copper or silver to give the calcines a market value; (*c*) as a waste product from roasting furnaces at a copper smelting plant.

Approximately three-fourths of a unit of sulphur is required on an average for each unit of manganese, iron, zinc, and lime dissolved. Most of the sulphur dioxide consumed in leaching is recovered in the final roasting of the sulphate and can be reused. Sulphur in combination with lime is lost. In view of the plant requirement, the relatively small size of the average western manganese deposit, and the uncertainty of the manganese market over a period of years, it would seem that with sulphur dioxide reduction of straight manganese ores must be confined to favorably situated copper smelting plants where sulphur dioxide of the proper strength is available as a waste product.

COMPLEX ORES.

The foregoing observations relate largely to ores in which manganese is the principal constituent. Some study was also given to the more complex manganiferous ores, such as silver ores, lead-silver ores, copper-silver ores, and zinc ores, containing sufficient manganese and occurring in sufficiently large quantities to warrant their consideration as a possible emergency source of manganese. Much laboratory work was done in an attempt to determine the relative solubility in sulphur dioxide of the various minerals that may be present in a complex ore, in the hope that some selective action might be discovered which would effect a separation between manganese and zinc. However, it was impossible to reproduce or to maintain in the drum the conditions that in the laboratory might produce certain preferential action on oxides over carbonates. Under operating conditions a large excess of acid is always present in some part of the drum; and the result is that the various SO_2 soluble manganese, zinc, and copper minerals are attacked practically simultaneously. If anything, zinc minerals (with the exception of crystalline zincite) will be dissolved fastest; the higher manganese oxides are next in the scale and the copper minerals last.

As the three minerals soluble in SO_2, copper, zinc, and manganese, are recovered together as sulphates in the filtrate, their separate recovery from sulphate solution was investigated. Copper offers no difficulty; it may be recovered by passing the solution over iron. In order to retain the manganese in solution during the precipitation of the copper, the specific gravity of the pregnant solutions in treating copper-bearing ores, must be lessened. The copper in solution will be replaced with iron which will, of course, result in a slightly lower manganese content in the sinter. Some preliminary work was done on the separation of zinc and manganese, which may be briefly summarized as follows:

(*a*) A straight oxidizing roast around 1,050° C. converts the sulphates to oxides. If these oxides are mixed with powdered coal and are then heated in a retort to 1,500° C., the zinc volatilizes with a recovery of 80 per cent of condensed metallic zinc.

(*b*) A 60 per cent zinc recovery as oxide is possible by subjecting the sulphates to a straight oxidizing roast until converted to oxides and then adding coal or other reducing agent and continuing the roast in the presence of air at 1,350° C.

(*c*) A chloridizing roast will cause the zinc to volatilize as chloride. If sodium chloride is used the soda fuses with the mass, thus preventing complete chloridization. If calcium chloride is used there is no fusion. The lime is converted to sulphate (insoluble in SO_2), part of the manganese becomes oxidized and part remains as sulphate. The latter is water-soluble and the oxidized manganese can be extracted by redissolving it with SO_2, an operation which calls for too much manipulation and expense to be commercially practicable except under very favorable conditions.

(*d*) Electrolysis does not seem practicable. The zinc goes to the cathode and the manganese to the anode. The H_2SO_4 set free at both poles attacks the zinc. Apart from this, there are the troubles engendered by foul solutions.

This investigation is incomplete and should receive further attention.

COSTS.

GENERAL ESTIMATE.

The cost of a 250-ton unit erected at the Salt Lake smelters or at the Douglas, Arizona smelter, would in normal times be roughly $100,000. A 500-ton unit should not cost to exceed $150,000. Under present (1918) conditions a 250-ton unit would cost perhaps as much as $200,000. The operating cost would be $12 to $14 per ton of shipping product based on an 80 per cent recovery of manganese from an ore which has a manganese content of 20 per cent. This includes crushing, leaching, and filtering 4 tons of ore; drying the filtrate and roasting the dried manganese sulphate. The resulting shipping product would be a sinter, free from silica, alumina, and phosphorus, which in the ores free from nonsulphide zinc or copper would contain 60 to 64 per cent manganese as oxides.

The cost of SO_2 treatment per ton of ore at a smelter where waste SO_2 gases are available should fall well within the estimated cost of $3.50 indicated in the following table.

Estimate cost of treating one ton of manganese ore.

Crushing in gyratory and ball mill	$0.40
Disposal of tailings	.10
Operation of drums and filtration plants	.15
Evaporation of $MnSO_4$ and roasting (heat, labor, and supplies)	[a] 1.50
Incidental handling	.25
Power	.30
Water	.10
General overhead	.20
Charge for SO_2	.50
	3.50

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[a] This item is high.

CALCULATED ESTIMATES FOR TYPICAL ORES.

The following estimates cover typical examples in Utah and Arizona. A railroad freight rate of $12 per long ton on shipment of sinter to eastern points has been assumed.

EXAMPLE 1.

Assume that a large siliceous ore body is located 10 miles from the main line of a railroad. The manganese content varies from 15 to 40 per cent, and the silica, 30 to 20 per cent, the balance being alumina, lime, and a little iron, with no zinc or copper. Average manganese content of the run-of-mine ore is 20 per cent, and all the manganese is soluble in SO_2.

Four tons of ore will (on an 80 per cent recovery basis) produce 1 short ton of high-grade sinter containing 60 per cent manganese, no silica, alumina, or phosphorus. The price, according to May, 1918, schedule is $1.30 per unit plus $6.50 silica premium = $84.50 per long ton, f. o. b. Chicago, or $84.50 − $12 freight = $72.50 per long ton f. o. b. smelters. The costs will be as follows:

Market value of one short ton shipping product, f. o. b. smelters, based on above schedule		$64.74
Cost of mining 1 ton of ore	$2.00	
Haulage to railroad at 30 cents per ton-mile	3.00	
Loading, sampling, switching and general handling	1.00	
Railroad freight to smelter	2.50	
SO_2 treatment charge	3.50	
SO_2 plant extinguishment	2.00	
Cost producing and treating 1 ton of ore	14.00	
Cost of producing 1 short ton of shipping product		56.00
Net profit on 4 tons of ore		8.74
Net profit per ton of ore mined		2.18

It is assumed that the market will hold for 12 months after completion of the SO_2 plant, or that 12 months' production has been contracted for, and that the entire cost of the plant (at war-time cost of $200,000) must be charged off against 100,000 tons of ore. The plant-extinguishment charge will therefore, be $2 per ton, as given in the table.

EXAMPLE 2.

Assuming that the shipping grade of the ore in example 1 could readily be raised to 25 per cent manganese at an additional mining cost of 50 cents per ton and to 30 per cent manganese at an additional cost of $1 per ton, the returns would be, respectively, as follows:

Four tons of 25 per cent ore will yield 1.19 long tons of 60 per cent manganese sinter, having a market value of [($84.50 − $12 freight)] × 1.19 = $86.30. The profit under these conditions would be [($86.30 − ($56 + 2)] ÷ 4 = $7.08 per ton of sorted ore at the mine.

Four tons of 30 per cent ore will yield 1.4 long tons of 60 per cent manganese sinter, having a market value of (\$84.50 − \$12 freight) ×1.4 = \$101.50 f. o. b. smelter. The profit would be [(\$101.50 − (\$56 + 4)] ÷ 4 = \$10.37 per ton of sorted ore at the mine.

Evidently both these sorted ores would stand a larger haulage charge.

EXAMPLE 3.

Assume that a manganiferous lead-silver ore has a lime-silica gangue with sufficient iron and lime to balance the silica after removal of the manganese. The manganese content is 20 per cent. This has a fluxing value of \$1.40. The removal of the manganese effects a shrinkage of smelter tonnage, which is in turn offset by a briquetting charge on the lixiviation tailings. The lead-silver contents are assumed to have a net value of \$4 per ton of original ore if the manganese is removed and \$6.40 per ton of original ore if smelted without removal of manganese.

Assuming the same general conditions and prices as obtained in example 1, it is evident that this would be a highly profitable wartime operation, as the \$4 smelter return per ton of original ore would be added to the \$2.18 profit on the manganese content per ton of ore mined, making a total profit of \$6.18 per ton of ore mined. An otherwise unprofitable operation would thus be made highly profitable and an important source of manganese would be developed.

EXAMPLE 4.

Assuming that under normal economic conditions there would be a ready eastern market for a 60 per cent manganese sinter at 75 cents per unit, the value of the manganese f. o. b. smelter would shrink to about \$28 and the maximum return per ton of original ore would be \$28 + \$16 = \$44.

Under normal conditions there would be an appreciable sealing down of operating costs and this operation would show a small profit as follows:

Market value at smelter of the manganese and other metals in four tons of ore		\$44.00
Mining per ton of ore	\$1.50	
Haulage to railroad at 20 cents per ton-mile	2.00	
Loading, sampling, and general handling	1.00	
Railroad freight to smelter	2.50	
SO_2 treatment charge	3.00	
SO_2 plant extinguishment and interest on investment (based on 4 years)	.50	
Cost of producing and treating 1 ton of ore	10.50	
Cost of mining and treating 4 tons of ore necessary to produce 1 short ton 60 per cent manganese shipping product		42.00
Profit on 4 tons or ore at mine		2.00
Profit on 1 ton of ore at mine		**.50**

A slight increase in either the manganese content or in the lead-silver content on the one hand or a decrease in transportation costs on the other hand would show a satisfactory balance.

POSSIBLE FUTURE APPLICATION OF SO_2 METHOD.

The war-need incentive to further investigation of this question by the Bureau of Mines no longer exists. It is probable, however, that there are certain properties where the foregoing information might be profitably applied, as these figures offer some encouragement when considered in connection with fairly large ore bodies which, in addition to the manganese, may contain other valuable minerals, such as (*a*) 1 to 2 per cent of copper (whether sulphide or nonsulphide); (*b*) sufficient lead or silver, or combined lead and silver to leave a residue that would pay to smelt; (*c*) sufficient zinc to warrant separating the manganese and the zinc at the initial treatment plant or to warrant experimentation along the lines of electric smelting and subsequent recovery of the zinc from the slag.

CHAPTER 6.—THE JONES PROCESS FOR CONCENTRATING MANGANESE ORES; RESULTS OF LABORATORY INVESTIGATIONS.[a]

By PETER CHRISTIANSON and W. H. HUNTER.

INTRODUCTORY STATEMENT.

During the summer of 1918 a "direct-reduction" process for treating manganese ores, known as the Jones process from the name of its inventor, John T. Jones, was investigated at the laboratory of the Minneapolis experiment station of the Bureau of Mines. This work was undertaken because the process offered possibilities for concentrating the manganese in the manganiferous iron ores of the Cuyana Range and elsewhere.

The importance of this work in relation to war needs has passed, but a record of the investigation may be of general interest to the industry.

The aim of the Jones process is to effect by metallurgical means a separation and consequent concentration of the manganese in ores in which the manganese is so intimately associated with the iron that ordinary methods of gravity or magnetic separation are impracticable.

The process consists of two stages. In the first, called the low-temperature reduction stage, two products are made, namely, (*a*) metallic iron, suitable for direct use in the manufacture of steel, and (*b*) slag. Most of the manganese is concentrated in the slag. Separation of these two products, as outlined by the inventor, is accomplished by grinding the sinter and passing it over a magnetic separation apparatus, producing thereby a high-grade magnetic iron concentrate, and a nonmetallic manganiferous sinter. This method was used in the preliminary tests. In the final test, however, separation was accomplished by pouring the material under treatment in a liquid condition and recovering the iron metal in the form of a button. This last method of separation, developed at the Minneapolis station of the bureau, has obvious advantages, and should the process become a commercial success, would doubtless be the method used.

The second stage, which may be called the high-temperature stage, involves smelting the manganiferous slag or sinter derived from the first stage, to produce a manganese alloy.

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[a] Prepared in cooperation with the school of mines of the University of Minnesota.

LOW-TEMPERATURE REDUCTION.

PRELIMINARY TESTS.

OUTLINE.

The ore used in preliminary tests was typical of those locally known as disseminated Cuyuna ores, containing approximately 24 per cent iron, 18 per cent manganese, and 26 per cent silica. The following temperatures were tried: 1050°, 1150°, 1250°, 1300°, and 1350° C.

The time was varied from one-half hour to three hours.

The reducing agent used was Elkhorn coal, containing 6.6 per cent ash and 40.8 per cent volatile matter; the amount was varied from 10 to 50 per cent of the ore.

APPARATUS USED.

The apparatus used in the tests were as follows:

Dixon graphite crucibles, size No. 2 for 200 grams of ore, and others in proportion to size of charge.

Gas muffle furnaces for temperatures of 1050° and 1150° C.; oil-fired muffles for 1250° C.; electric carbon resistor furnace for 1300° and 1350° C.

Pyrometers: Hoskins base-metal couples and Hoskins meters for temperatures up to and including 1250° C.

Platinum-and-platinum-rhodium couples and Leeds & Northrup potentiometer for temperatures above 1250° C.

Hand-crushing apparatus, screens, hand magnets, and panning apparatus for separating magnetic from nonmagnetic material.

The temperatures indicated are those of the muffle outside the crucibles. The temperatures of the charges within the crucibles were probably nearly the same, except as regards those charges heated only 30 minutes. Temperatures of heats above 1250° C. were taken within the charge. Readings were generally taken every five minutes.

PROCEDURE.

CHARGING AND HEATING.

When muffle furnaces were used the ore and the reducing agent, crushed to pass 10 mesh, were mixed and charged into cold crucibles; these were placed in the muffle heated to nearly the desired temperature. When crucible furnaces were used the charge was put into crucibles heated to nearly the desired temperature. By using the muffle furnaces a number of crucibles could simultaneously be subjected to practically the same temperatures, thus keeping this variable constant; and a variation in time could be obtained by removing the crucibles at the end of different periods of time.

DISCHARGING.

At the end of the reduction period the contents of the crucibles were discharged into cast-iron molds, so as to effect a rapid cooling.

SEPARATION.

After cooling, the content of each crucible was generally separated into the following:

$$\left.\begin{array}{l}\text{1. Metal} + 30 \text{ mesh}\\ \text{2. Metal} - 30 \text{ mesh}\end{array}\right\}\text{magnetic part.}$$

$$\left.\begin{array}{l}\text{3. Slag}\\ \text{4. Carbon.}\end{array}\right\}\text{nonmagnetic part.}$$

This separation was effected by a series of crushings, screenings, and magnetic separations, frequently supplemented by panning. Each of these parts, if panned, was weighed and sampled for analysis.

ANALYSES.

The determinations generally made were: Per cent soluble iron; per cent total iron; per cent soluble manganese; per cent total manganese; per cent insoluble.

CALCULATIONS.

The results obtained from the analyses, together with the weights of the various parts, were combined by calculations to obtain the metal content and percentages of iron and manganese recovered in each of total magnetic and nonmagnetic parts.

The following formulas were used in calculations:

Formulas used in calculating recoveries.

Percentage Extraction.

For magnetic part.

Grams Fe.

Grams of magnetic part+30×per cent Fe content=.................. ....

Grams of magnetic part−30×per cent Fe content=.................. ....

Sum=total grams Fe in magnetic part=............................ ....

$$\frac{\text{Grams Fe in magnetic part}\times 100}{\text{Grams Fe in charge}}=\text{per cent extraction of Fe.}$$

For nonmagnetic part.

Grams Fe.

Grams of slag×per cent of Fe content=................................ ....

Grams of carbon×per cent of Fe content=............................ ....

Sum=total grams Fe in nonmagnetic part=......................... ....

$$\frac{\text{Grams Fe in nonmagnetic part}}{\text{Grams Fe in charge}}\times 100=\text{per cent extraction Fe.}$$

Recovery Per 100 Units of Ore.

Per cent Fe or Mn in ore×per cent extraction=Recovery per 100 units of ore.

Per Cent Soluble Iron in Concentrates.

Grams soluble Fe.

Per cent soluble Fe×grams of magnetic part+30=...................... ....

Per cent soluble Fe×grams magnetic part−30=......................... ....

Sum=Total grams soluble Fe in magnetic part=.................... ....

$$\frac{\text{Sum}\times 100}{\text{Grams of magnetic part}}=\text{per cent soluble Fe in magnetic part.}$$

Per Cent Soluble Iron in Soluble Part of Magnetic Part.

	Grams soluble matter.
Per cent soluble matter × magnetic + 30 = ...	....
Per cent soluble matter × magnetic − 30 = ...	....
Sum = Total grams soluble matter in magnetic part = ...	....

$$\frac{\text{Total grams soluble Fe in magnetic part}}{\text{Grams soluble matter in magnetic part}} \times 100 = \text{per cent soluble Fe in soluble part of magnetic part.}$$

Results obtained in the tests are given in Table 11.

TABLE 11.—*Results of preliminary low-temperature reduction tests with the Jones process.*

RESULTS WITH VARYING TEMPERATURES.

(*a*) DATA.

Test No.	16-D.	17-D.	35-D.	31.	32.
Temperature, °C.	1,050	1,150	1,250	1,300	1,350
Ore, grams	363	363	160	363	363
Coal, grams	91	91	40	91	91
Time, minutes	120	120	120	120	120

(*b*) EXTRACTION.

Temperature °C.	Product.	Weight, grams.	Iron.		Manganese.	
			Total, per cent.	Recovery, per cent.	Total, per cent.	Recovery, per cent.
	Ore	363.0	24.36	....	18.12	....
1,050	Magnetic part	233.5	32.0	84.7	19.8	70.5
1,050	Nonmagnetic part	84.6	5.7	5.5	19.3	24.8
1,150	Magnetic part	131.4	58.5	86.9	15.0	30.0
1,150	Nonmagnetic part	189.4	8.18	17.5	20.9	60.2
[a] 1,250	Magnetic part	40.5	90.6	93.9	3.2	4.5
[a] 1,250	Nonmagnetic part	85.3	2.0	4.4	28.9	84.8
1,300	Magnetic part	94.8	87.2	93.5	7.0	10.0
1,300	Nonmagnetic part	209.0	2.4	5.7	26.2	83.0
1,350	Magnetic part	96.6	80.4	91.0	9.4	14.3
1,350	Nonmagnetic part	202.2	4.4	10.0	23.7	72.8

[a] In test 35-D, 160 grams of ore was used.

(*c*) RECOVERY.

Test No.	Variable temperature, °C.	Iron.			Manganese.		
		Grams in 100 grams of ore.	Recovery.		Grams. in 100 grams of ore.	Recovery.	
			Per cent.	Grams.		Per cent.	Grams.
16-D	1,050	24.36	84.7	20.5	18.12	70.5	12.60
17-D	1,150	24.36	86.9	21.1	18.12	30.0	5.44
35-D	1,250	24.36	93.9	22.7	18.12	4.5	0.80
31	1,300	24.36	93.5	22.6	18.12	5.7	1.03
32	1,350	24.36	91.0	22.0	18.12	14.3	2.60

RESULTS WITH VARIATION OF TIME, AT 1,150° C.

(*a*) DATA.

Test No.	17-A.	17-B.	17-C.	17-D.	17-E.
Time, minutes	30	60	90	120	150
Ore, grams	363	363	363	363	363
Coal, grams	91	91	91	91	91
Temperature, °C.	1,150	1,150	1,150	1,150	1,150

Results with Variation on Time, at 1,150° C.—Continued.

(b) EXTRACTION.

Minutes.	Product.	Weight, grams.	Iron.		Manganese.	
			Total, per cent.	Recovery, per cent.	Total, per cent.	Recovery, per cent.
	Ore	363.0	24.36	....	18.12	....
30	Magnetic part	156.0	38.7	68.4	15.8	37.5
30	Nonmagnetic part	163.0	10.0	18.6	18.8	34.8
60	Magnetic part	204.4	38.4	89.0	17.7	55.5
60	Nonmagnetic part	125.5	7.1	10.1	21.3	40.6
90	Magnetic part	166.0	44.8	84.3	18.7	47.0
90	Nonmagnetic part	154.2	8.4	14.6	22.7	53.3
120	Magnetic part	131.4	58.5	86.9	15.0	30.0
120	Nonmagnetic part	189.4	8.18	17.5	20.9	60.2
150	Magnetic part	122.7	57.0	79.3	13.6	25.2
150	Nonmagnetic part	178.5	8.1	16.4	22.1	59.9

(c) RECOVERY.

Test No.	Variable, minutes.	Iron.			Manganese.		
		Grams per 100 grams of ore.	Recovery.		Grams per 100 grams of ore.	Recovery.	
			Per cent.	Grams.		Per cent.	Grams.
17-A	30	24.36	68.4	16.6	18.12	37.5	6.8
17-B	60	24.36	89.0	21.7	18.12	55.5	10.1
17-C	90	24.36	84.3	20.5	18.12	47.0	8.5
17-D	120	24.36	86.9	21.2	18.12	30.0	5.4
17-E	150	24.36	79.3	19.1	18.12	25.2	4.6

(d) SOLUBLE IRON IN MAGNETIC PART.

Minutes.	Product.	Weight, grams.	Total, grams.	Percentage of soluble iron	Grams of soluble iron.	Total grams of soluble iron.	Percentage of soluble iron.
30	Magnetic part +30	....	....	....	....	....	....
	−30	....	....	....	....	....	....
60	Magnetic part +30	131.5	204.4	46.0	60.5	77.2	37.7
	−30	72.9		22.9	16.7		
90	Magnetic part +30	135.1	166.0	47.8	64.7	72.6	43.6
	−30	30.9		25.4	7.9		
120	Magnetic part +30	84.15	131.4	62.8	52.8	68.4	52.0
	−30	47.2		33.0	15.6		
150	Skull ..	50.9	122.7	44.0	22.4	66.6	54.4
	Powder ..	71.8		61.6	44.2		

(e) SOLUBLE IRON IN SOLUBLE PART OF MAGNETIC PART.

Minutes.	Product.	Weight, grams.	Total grams.	Percentage of soluble matter.	Grams of soluble matter.	Total grams of soluble matter.	Total grams of soluble iron.	Percentage of soluble iron in soluble matter.
30	Magnetic part +30	....	....	....	....	....	....	....
	−30	....	....	....	....	....	....	....
60	Magnetic part +30	131.5	204.4	75.5	98.8	144.4	77.2	53.4
	−30	72.9		62.2	45.6			
90	Magnetic part +30	135.1	166.0	73.0	98.8	117.4	72.6	61.5
	−30	30.9		59.3	18.6			
120	Magnetic part +30	84.15	131.4	81.7	68.8	99.1	68.4	69.7
	−30	47.2		64.2	30.3			
150	Skull ..	50.9	122.7	67.2	34.2	91.0	66.6	73.4
	Powder ..	71.8		79.2	56.8			

RESULTS WITH VARIATION OF TIME, AT 1,250° C.

(a) DATA.

Test No.	35-A.	35-B.	35-C.	35-D.	35-E.
Time, minutes	30	60	90	120	150
Ore, grams	160	160	160	160	160
Coal, grams	40	40	40	40	40
Temperature, °C	1250	1250	1250	1250	1250

(b) EXTRACTION.

Minutes.	Product.	Weight, grams.	Iron.		Manganese.	
			Total per cent.	Recovery per cent.	Total per cent.	Recovery per cent.
	Ore	160.0	24.36	....	18.12	....
30	Magnetic part	67.4	38.26	66.2	16.7	38.6
30	Nonmagnetic part	78.7	10.8	21.8	20.5	55.5
60	Magnetic part	44.0	79.2	89.2	4.8	7.3
60	Nonmagnetic part	92.0	3.3	7.7	27.4	86.9
90	Magnetic part	39.7	88.6	90.4	4.3	5.9
90	Nonmagnetic part	93.8	4.9	11.8	26.3	85.2
120	Magnetic part	40.5	90.6	93.9	3.2	4.5
120	Nonmagnetic part	85.3	2.0	4.4	28.9	84.8
150	Magnetic part	42.7	91.0	98.5	4.0	5.9
150	Nonmagnetic part	81.7	1.6	3.3	29.0	81.7

(c) RECOVERY.

Variable, minutes.	Iron.			Manganese.		
	Grams per 100 grams of ore.	Recovery.		Grams per 100 grams of ore.	Recovery.	
		Per cent.	Grams.		Per cent.	Grams.
30	24.36	66.2	16.1	18.12	38.6	7.1
60	24.36	89.2	21.7	18.12	7.3	1.3
90	24.36	90.4	22.0	18.12	5.9	1.1
120	24.36	93.9	22.8	18.12	4.5	0.8
150	24.36	99.5	24.2	18.12	5.9	1.1

(d) SOLUBLE IRON IN MAGNETIC PART.

Minutes.	Products.		Weight, grams.	Total, grams.	Percentage of soluble iron.	Grams of soluble iron.	Total grams of soluble iron.	Percentage of soluble iron.
30	Magnetic part	+30	....	....	....	....	....	....
		−30	....	....	....	....		
60	Magnetic part	+30	34.3	44.0	81.0	27.8	34.8	79.0
		−30	9.7		71.8	7.0		
90	Magnetic part	+30	33.7	39.7	90.8	30.6	35.2	88.6
		−30	6.0		77.0	4.0		
120	Magnetic part	+30	36.4	40.5	92.3	33.6	36.6	90.5
		−30	4.1		71.95	3.0		
150	Skull	..	39.1	47.7	91.5	35.8	38.4	9.00
	Powder	..	3.6		71.75	2.6		

RESULTS WITH VARIATION OF TIME, AT 1,250° C.—Continued.

(*e*) SOLUBLE IRON IN SOLUBLE PART OF MAGNETIC PART.

Minutes.	Product.	Weight, grams.	Total, grams.	Percentage of soluble matter.	Grams of soluble matter.	Total grams of soluble matter.	Total grams of soluble iron.	Percentage of soluble iron in soluble matter.
30	Magnetic part +30	....	....	....	....	....	....	....
	Magnetic part −30	....		....	....			
60	Magnetic part +30	34.3	44.0	90.0	30.9	39.4	34.8	88.5
	Magnetic part −30	9.7		88.0	8.5			
90	Magnetic part +30	33.7	39.7	98.2	33.1	38.5	35.2	91.5
	Magnetic part −30	6.0		90.0	5.4			
120	Magnetic part +30	36.4	40.5	100.0	36.4	40.0	36.6	91.2
	Magnetic part −30	4.1		87.8	3.6			
150	Magnetic part +30	39.1	42.7	99.6	39.0	42.2	38.4	91.0
	Magnetic part −30	3.6		87.7	3.2			

RESULTS WITH VARIATION OF AMOUNT OF REDUCING AGENTS.

(*a*) DATA.

Test No.	34-A.	34-B.	34-C.	34-D.
Coal, grams	20	40	60	80
Ore, grams	200	200	200	200
Temperature, °C	1,250	1,250	1,250	1,250
Magnetic part, grams	64.6	53.5	50.1	52.7
Nonmagnetic part, grams	94.2	109.7	121.6	127.6

(*b*) EXTRACTION.

Coal, per cent of ore.	Product.	Weight, grams.	Iron.		Manganese.	
			Total, per cent.	Recovery, per cent.	Total, per cent.	Recovery, per cent.
	Ore	200.0	24.36	....	18.12	....
10	Magnetic part	64.6	68.3	90.5	10.7	19.0
10	Nonmagnetic part	94.2	6.1	11.7	27.4	71.3
20	Magnetic part	53.5	90.0	98.8	4.7	13.0
20	Nonmagnetic part	109.7	3.6	7.4	28.3	85.8
30	Magnetic part	50.1	89.9	92.5	4.8	6.5
30	Nonmagnetic part	121.6	4.1	10.2	26.0	87.2
40	Magnetic part	52.7	70.5	76.2	13.5	19.6
40	Nonmagnetic part	127.6	10.7	27.9	21.1	74.3

(*c*) RECOVERY.

Coal, percentage (by weight) of ore.	Iron.			Manganese.		
	Pounds per 100 pounds of ore.	Recovery.		Pounds per 100 pounds of ore.	Recovery.	
		Per cent.	Pounds.		Per cent.	Pounds.
10	24.36	90.5	22.0	18.12	19.0	3.5
20	24.36	98.8	24.0	18.12	7.2	1.3
30	24.36	92.5	22.5	18.12	6.5	1.2
40	24.36	76.2	18.5	18.12	19.6	3.6

The foregoing results have been plotted in figures 2 to 9.

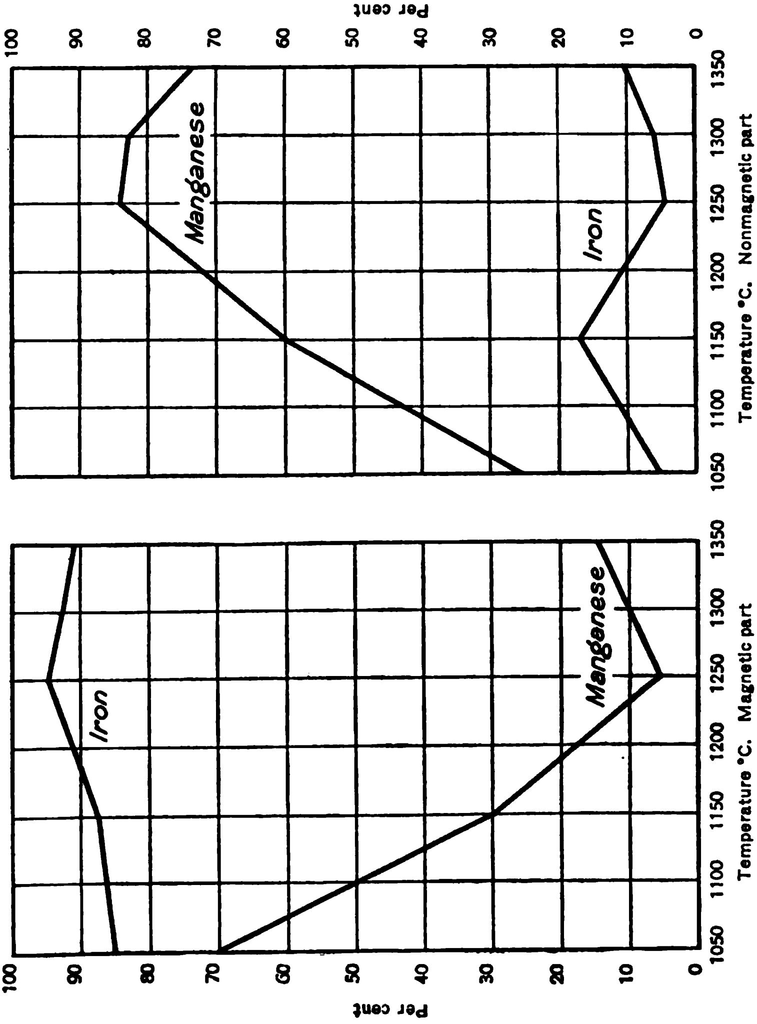

FIGURE 2.—Curves showing recoveries of iron and of manganese in heats 16, 17, 32, and 35, with varying temperatures.

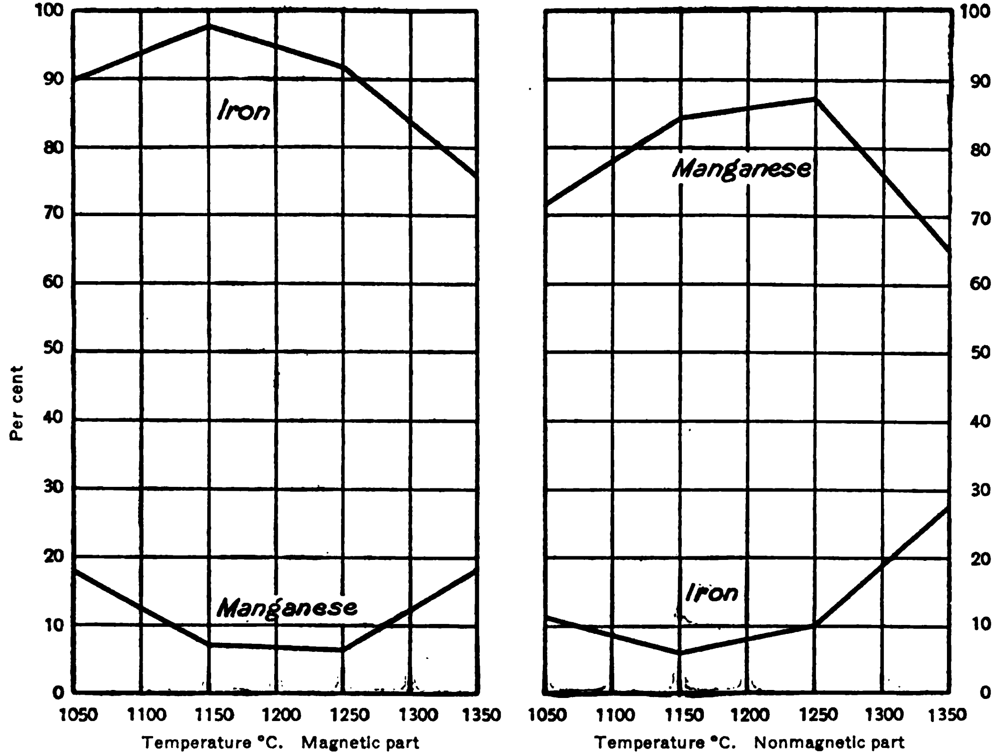

FIGURE 3.—Curves showing recoveries of iron and of manganese in heat 34 at 1,250° C. and with varying quantities of reducing agent.

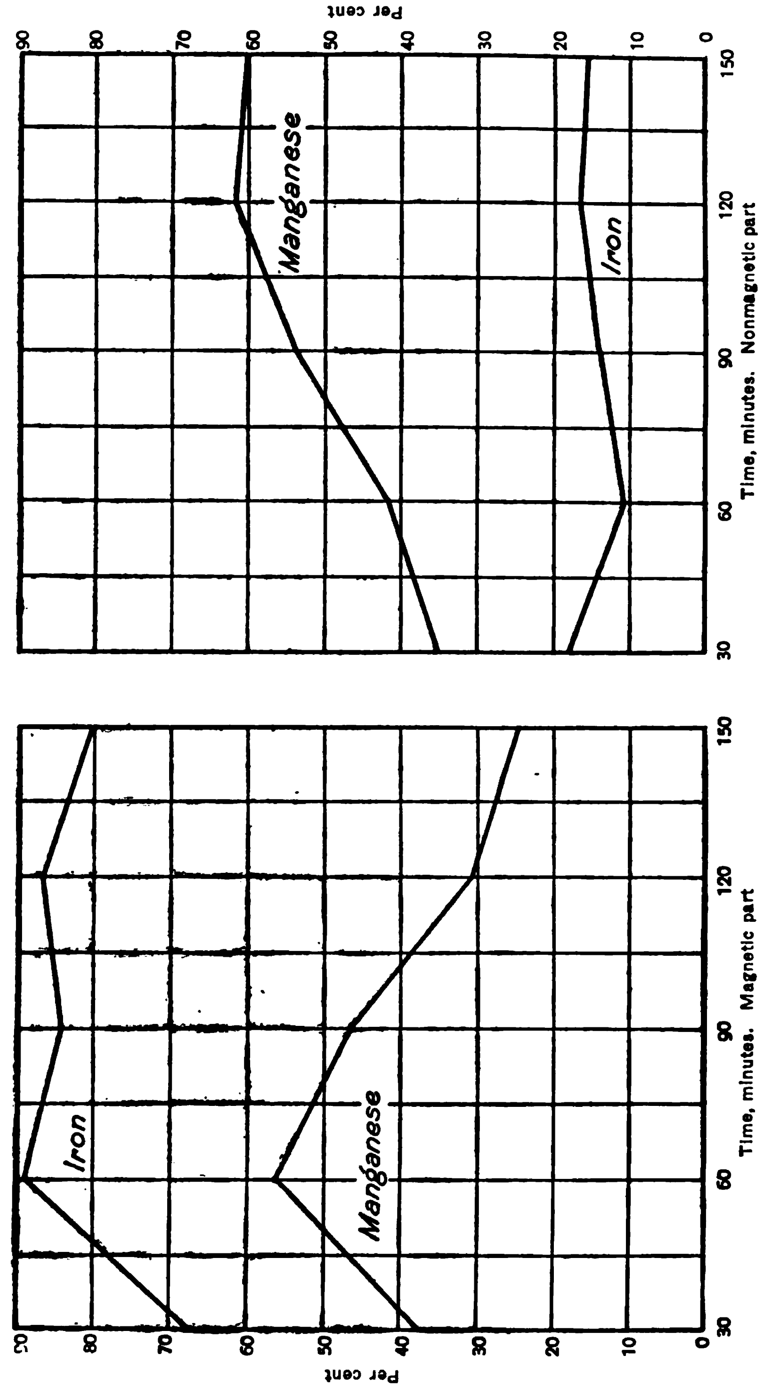

FIGURE 4.—Curves showing recoveries of iron and of manganese in heat 17 at 1,150° C. and with varying time.

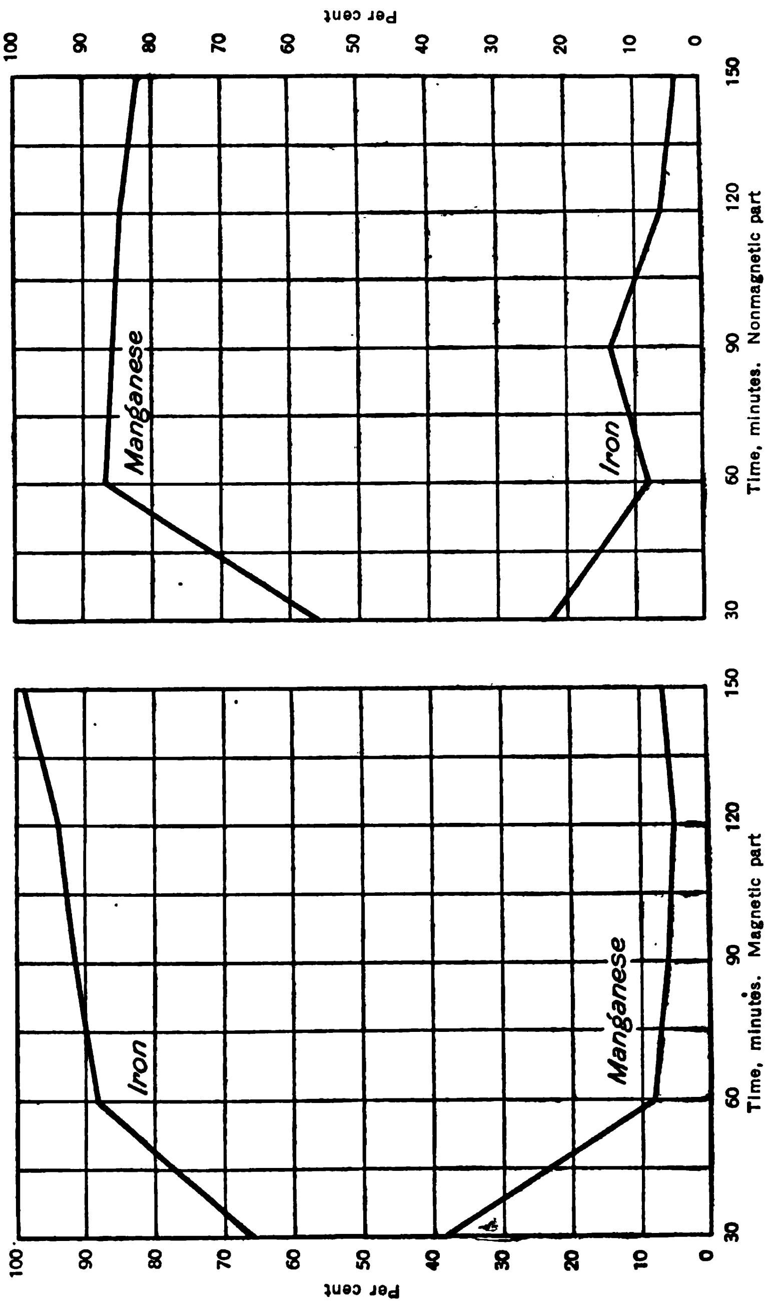

FIGURE 5.—Curves showing recoveries of iron and of manganese in heat 35 at 1,250° C. and with varying time.

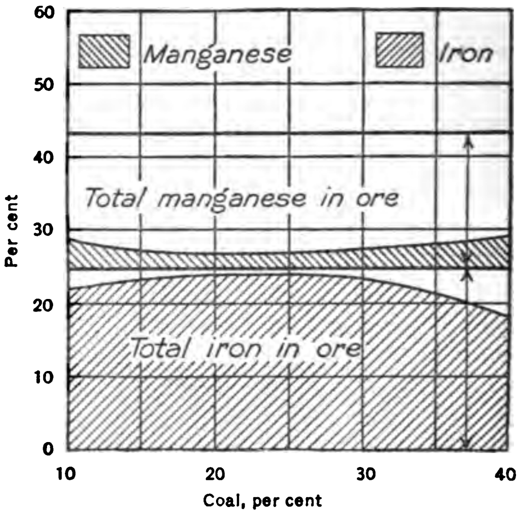

Figure 6.—Curves showing percentages of iron and of manganese in magnetic part in heat 34 at 1,250° C. and with varying reducing agent.

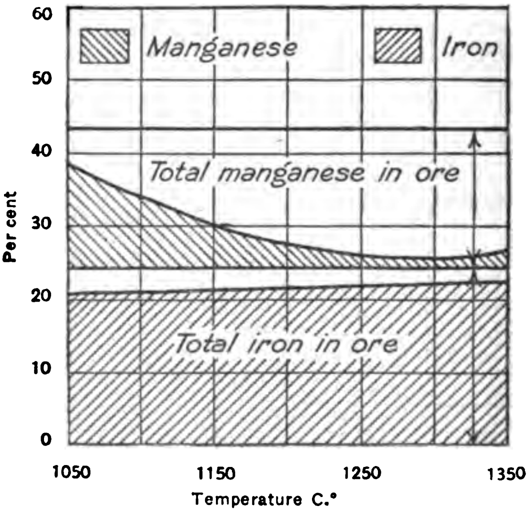

Figure 7.—Curves showing percentages of iron and of manganese in magnetic part in heats 16, 17, 31, 32, and 35 with varying temperatures.

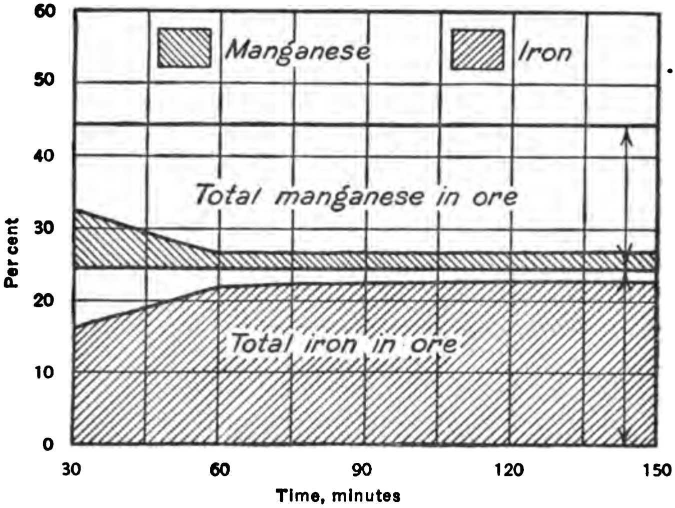

Figure 8.—Curves showing percentages of iron and of manganese in magnetic part in heat 35 at 1,250° C.

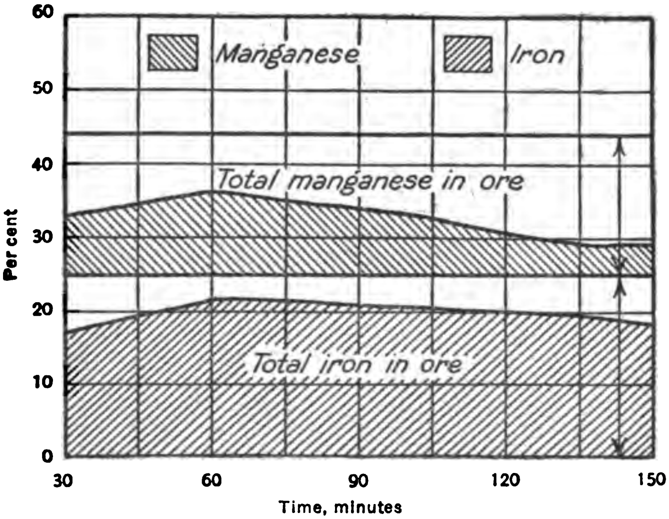

Figure 9.—Curves showing percentages of iron and of manganese in magnetic part in heat 17 at 1,150 °C.

CONCLUSIONS.

Figures 2 to 9 indicate that the following are the requisite conditions for the low-temperature reduction of the iron:

(*a*) Time, preferably 30 to 40 minutes, although satisfactory results were obtained up to 2 hours.

(*b*) Temperature, 1,250° C.

(*c*) Reducing agent, 20 per cent coal.

Analyses of ores used in final tests with low-temperature reduction.

Ore.	Fe.	Mn.	SiO_2.	P.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
West End	24.36	18.12	26.50	0.097
Ferro	28.55	24.55	17.60	.091
1019 E	35.15	13.68	9.46	.179
1016 Cts	40.30	13.08	10.06	.107
381 Cts	35.40	10.25	21.36	.059

Two kilograms of each ore, crushed to 10 mesh and mixed with 20 per cent reducing agent, was charged into a large graphite crucible heated to redness.

The temperature was raised to 1,250° C. and kept as closely as possible between 1,250° and 1,275° C. until reaction was complete.

The time was varied to some extent according to the appearance of the charge, with a maximum time of two and one-half hours, except in heat 43.

The reducing agent used was Elkhorn coal like that used in the preliminary tests, except that in heat 41 coke was used.

The furnace used was a Case oil-fired melting furnace No. 40.

The method of handling the furnace products was practically the same as in the preliminary tests. However, the entire contents of the crucible were poured in a liquid condition, and most of the metal produced was recovered in the form of a large button.

TABULATED RESULTS.

Detailed results of the tests are presented in Table 12 following:

TABLE 12.—*Results of final tests with low-temperature reduction.*

Test No.	36	37	38	39	40	41	42	43	44	45	46
Charge.											
Weight:											
Ore.....grams	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000
Coal.....do	500	400	400	500	400	300	400	400	400	400	400
Lime.....do	......	......	......	......	......	......	......	......	......	......	100
Ore analysis:											
Fe.....per cent	24.36	28.55	(a)	35.15	(b)	(b)	40.30	(c)	(b)	35.40	(d)
Mn.....do	18.12	24.55	(a)	13.68	(b)	(b)	13.08	(c)	(b)	10.25	(d)
SiO_2.....do	26.50	17.60	(a)	9.46	(b)	(b)	10.06	(c)	(b)	21.36	(d)
P.....do	0.097	0.091	(a)	0.179	(b)	(b)	0.107	(c)	(b)	0.059	(d)
Metal.											
Weight.....grams	463	591	571	927	697	683.60	781	818	703	691	695
Percentage, by weight of ore	24.15	29.55	28.55	36.35	34.85	34.18	39.05	40.90	35.15	34.55	34.75
Analysis:											
Fe.....per cent	92.8	88.90	95.20	95.00	95.90	92.80	96.30	94.50	94.50	94.70	95.60
Mn.....do	2.64	3.90	0.70	1.25	0.56	0.45	0.28	0.84	0.53	0.46	0.35
P.....do	0.4	0.25	0.25	......	0.45	0.50	0.24	0.25	0.46	0.184	0.184
S.....do	0.04	0.003	0.035	0.043	0.057	0.071	0.052	0.027	0.054	0.095	0.079
Recovery:											
Fe.....per cent	87.8	91.8	95.2	98.3	95.0	90.3	93.3	96.0	94.5	93.0	93.6
Mn.....do	3.34	4.7	0.8	3.33	1.4	1.1	0.8	2.6	1.4	1.6	1.2
P.....do	95.0	80.2	80.2	......	81.6	82.7	82.0	93.5	85.0	100.0	100.0
Slag.											
Weight.....grams	1,137	1,056	1,107	851	816	930	785	819	905	955	967
Percentage, by weight of ore	56.9	52.8	55.35	42.55	40.8	46.5	39.3	40.9	45.3	47.7	48.4
Analysis:											
Fe.....per cent	4.9	3.9	2.9	2.2	4.4	6.8	6.4	2.6	4.5	4.6	4.2
Mn.....do	27.3	40.0	42.7	29.2	29.55	26.0	30.8	28.7	27.75	20.6	18.75
Recovery:											
Fe.....do	11.4	7.3	5.6	2.6	5.1	9.0	6.2	2.6	5.8	6.2	5.7
Mn.....do	85.5	86.0	95.0	90.7	88.2	88.3	92.4	89.8	93.5	95.7	88.4

[a] Same as for No. 37.

[b] Same as for No. 39.

[c] Same as for No. 42.

[d] Same as for No. 45.

DISCUSSION.

The foregoing tabulation is self-explanatory. The results were obtained practically under the conditions indicated under preliminary tests. However, the appearance of the charge, and in particular the condition of the slag and metal, were also used to determine the end of the reaction.

PRODUCTS.

On averaging all the results 93.5 per cent of the iron and 2.0 per cent of the manganese contained in the ore were reduced and entered the metal, while 4.3 per cent of the iron and 90.3 per cent of the manganese remained in the slag. The composition of the metal obtained in all the tests varied as follows:

Iron, per cent, 89.9 to 96.3; average, 94.2.

Manganese,[a] per cent, 0.28 to 3.9; average, 1.0.

Phosphorus, per cent, 0.1842 to 0.50, or 88.0 per cent of that contained in the ore.

Sulphur, per cent, 0.003 to 0.097, or about 10 per cent of that contained in the charge.

ORES.

All the ores tested were of the intimately disseminated Cuyuna manganiferous type and represent a wide range of iron and manganese content. All were crushed to pass a 10-mesh screen, except that used in tests 45 and 46, which was crushed to 8-mesh. It is probable that if coarser material were used the time necessary to complete the reaction would increase. The uniform results obtained in testing such a variety of ores would tend to show that this low-temperature reduction is applicable to a much greater variety of ores.

TIME.

A considerable range of time at which the charge was kept at a temperature between 1,250° and 1,300° C. was used. From the data thus obtained it is apparent that, under the conditions given, the best results are obtained when the charge is kept at this temperature for 30 to 40 minutes.

REDUCING AGENT.

Elkhorn coal crushed to 10 mesh was used in all tests, except No. 41, in which foundry coke, crushed to the same mesh, was used. The Elkhorn coal analyzed 6.66 per cent ash, 40.98 per cent volatile matter, and 0.645 per cent sulphur. The coke analyzed 13.9 per cent ash, 3.33 per cent volatile matter, and 1 per cent sulphur.

No difference was noticed in the reaction when coke was used, except that the slag fused at a lower temperature. This was probably due to the larger percentage of ash in the coke. The use of coke also tends to increase the sulphur in the metal. The proportion required did not seem to vary with the character of the ore.

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[a] Discarding the manganese content of metal obtained in tests 36 and 37, the reactions of which were not completed, the average manganese content of the metal would be reduced to 0.60 per cent.

LIME.

The addition of lime in test 46 did not materially change the results. Its use as a flux would probably be necessary to obtain a high recovery of iron from ores containing small amounts of manganese and large percentages of silica. The lime used was of the commercial air-slaked type.

FINAL TESTS.

OUTLINE.

In the final tests with low-temperature reduction, ores of the following analyses were used:

HIGH-TEMPERATURE REDUCTION TESTS.

GENERAL OUTLINE.

The slag obtained by the process of low-temperature reduction was crushed to pass a 30-mesh screen, mixed with 25 per cent coke crushed to pass a 10-mesh screen, and charged into a carbon crucible which was heated in an ordinary electric carbon resistor furnace.

CHARGE.

An analysis of the slag used for the charge was: Iron, 4.9 per cent; manganese, 28.68 per cent; SiO_2, 31.1 per cent; Al_2O_3, not determined. An analysis of the coke used as a reducing agent is given above. Slag, coke, and lime were mixed before putting the charge into the crucible. The only variable used in these tests was lime. This was air-slaked material similar to that used in the low-temperature final tests.

PYROMETER.

The temperature was measured by means of a Scimatco pyrometer. This instrument, though not very satisfactory, was the best available. It was first calibrated against a Hoskins thermocouple, having a temperature capacity up to 1,100° C. With great care in observing by keeping the comparison lamp at a constant amperage fairly concordant results were obtained. The temperature measured was that of the surface of the charge only, but when a charge is in constant agitation, owing to the carbon monoxide produced by the reaction, it is assumed that the surface temperature serves as a fair indication of the temperature of the crucible contents.

CRUCIBLES.

A number of standard crucibles were tried, as follows: Ordinary assay crucible, graphite crucible, and alundum crucible. None of these could withstand the conditions. The crucibles finally used were cut on a lathe from 4-inch carbon electrodes. With care they would hold 200 grams of crushed slag mixed with the requisite coke and lime. These crucibles suffered deterioration only from oxidation at the upper edge and could be used until they became too shallow to hold the charge.

TEMPERATURE.

The temperature at which a satisfactory reaction took place was found to be about 1,450° C. This satisfactory reaction may be designated as uniform and rapid. Above 1,450° C. the reaction was so rapid that it caused violent boiling, and below this temperature the reaction was so slow and inactive that no appreciable metal was reduced. Hence, the condition of the reaction was used as a criterion for temperature regulation.

TABULATED RESULTS.

The detailed results of the high-temperature reduction tests are presented in the tabulation following:

TABLE 13.—*Results of high-temperature reduction tests.*

Test No.	54	55	56	57	58
Charge.					
Weight:					
Slag ... grams	200	200	200	200	200
Coke ... do	50	50	50	50	50
Lime ... do	40	30	20	30	00
Analysis:					
Fe ... per cent	4.9	(a)	(a)	(a)	(a)
Mn ... do	28.68	(a)	(a)	(a)	(a)
SiO_2 ... do	31.10	(a)	(a)	(a)	(a)
Alloy produced.					
Weight ... grams	66.0	66.3	66.7	76.4	81.5
Percentage of slag charged	33.0	33.15	33.35	37.7	40.75
Analysis:					
Fe ... per cent	16.8	15.40	16.40	12.5	12.0
Mn ... do	66.9	65.00	6.42	62.4	58.4
Si ... do	10.7	13.77	13.56	23.6	25.0
Recovery, per cent of content of slag charged:					
Fe	[b] 112.0	[b] 102.6	[b] 106.0	[b] 97.0	[b] 100.0
Mn	77.2	75.4	74.8	83.2	84.0
Si	24.1	31.4	31.0	62.0	70.5
Slag produced.					
Weight ... grams	134.0	108.7	112.3	83.3	82.9
Percentage of slag charged	67.0	54.4	56.15	36.9	41.45
Analysis:					
Fe ... per cent	1.28	1.94	2.15	1.37	1.30
Mn ... do	6.15	4.25	7.90	4.70	5.44
SiO_2 ... do	36.25	34.85	37.80	38.10	30.60

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[a] See test 54 for analysis.

[b] Recoveries are calculated on content of slag charged. Discrepancy shown by getting more than 100 per cent recovery is due to fact that coke contains iron as part of the ash.

DISCUSSION.

GENERAL.

Coke was used as a reducing agent because it was thought to be the best commercial source of carbon suitable for this high-temperature reduction. It was mixed well with the rest of the charge and seemed to remain more or less disseminated through the charge after the slag portion of the charge had melted. The amount used in the tests does not indicate the proportions required on a commercial scale where no other source of carbon would be present, because with the use of carbon crucibles there was always an excess of carbon present from this source.

Lime was added to the charge to render the resulting slag sufficiently liquid to pour. The amount added was varied for the purpose of controlling the reduction of silicon. However, the effect of lime in this respect can not be considered very definite, although the tendency is to decrease the silicon in the alloy as the lime is increased in the charge.

The data obtained indicate that the reduction of silica to metallic silicon increases as the temperature is increased and is a function both of the temperature and of the lime added. Possibly, if the lime were incorporated in the slag portion of the charge, its effect would be more pronounced.

ALLOY.

The metal produced is essentially a silico-ferro-manganese, consisting of about 15 per cent iron, 60 to 67 per cent manganese, and 10 to 25 per cent silicon. The percentage of silicon increases with the temperature of the reaction; and as the silicon increases the percentage of manganese in the alloy will correspondingly decrease.

RECOVERY.

The excess of iron recovered, together with some iron remaining in the resulting slag, is explained by sources of iron being present in the charge, which were not taken into account in making the calculations. The recovery is based on the slag portion of the charge only. Under the given conditions, the manganese remaining in the resulting slag was not less than about 5 per cent. It is, therefore, evident that as the slag volume is decreased the recovery of manganese is increased. This is shown in experiments Nos. 57 and 58. The recovery of manganese, as given in the tabulated results, varied from 75 to 84 per cent, and averaged 79 per cent.

SLAG.

As the iron and manganese is reduced from the charge, the fusibility of the slag decreases and its viscosity increases as a result of the increase in unbalanced silica. The presence of lime in the charge is advantageous, increasing the fusibility of the slag, and decreasing its viscosity, so that it may be handled in a liquid condition.

SUMMARY.

The essential results contained in the foregoing report may be summarized in semitabular form as follows:

Summary of results of experiments with the Jones process.

LOW-TEMPERATURE REDUCTION, 1,250° TO 1,300° C.

Charge:

Reducing agent, 20 per cent of a good grade of coal.

Ore:

Fe, per cent	24.0 to 40.0
Mn, per cent	10.0 to 24.0
SiO_2, per cent	10.0 to 26.0
Al_2O_3, per cent	Not determined.

Products:

	Approximate composition, per cent.	Recovery, per cent.
Liquid metal:		
Fe,	94.00	94.00
Mn,	1.00	2.0
Si,	.10 (maximum)	.....
P,	.50 (maximum)	88.0 (average)
Liquid slag:		
Fe,	5.00	6.0
Mn,	29.00	90.0
SiO_2,	31.00	.....
Loss	Manganese	8.0

HIGH-TEMPERATURE REDUCTION, 1,450° C.

Charge:

Liquid slag from low-temperature reduction.

Coke, 25 per cent of slag.

Products:

	Approximate composition, per cent.	Recovery, per cent.
Liquid metal:		
Fe	15	100.0
Mn	55 to 65	80.0
Si	10 to 25	.....
Liquid slag:		
Fe	1.0 to 2.0	
Mn	4.0 to 6.0	
SiO_2	30.0 to 40.0	

CONCLUSIONS.

The foregoing results seem to indicate that by the Jones process concentration of manganese in the finely disseminated manganiferous iron ores is metallurgically possible. A discussion of the commercial possibilities of the process will not be attempted.

The metal produced in the low-temperature reduction is suitable for conversion into steel by any basic steel process. It could not be used as a foundry metal. Whether the low-temperature reduction could be modified to yield a foundry metal would require further experiments. By a suitable regulation of temperature and by the use of iron ore in the place of manganiferous ores, it is possible that a foundry metal could be produced.

The high-temperature reduction produces an alloy that may be termed a silico-ferromanganese. Although an alloy of this nature would be new to the steel industry, it could probably be used to advantage. About 72 per cent of the manganese in the ore appears in the alloy.

CHAPTER 7.—COST OF PRODUCING FERRO-GRADE MANGANESE ORES.

By C. M. Weld and W. R. Crane.

INTRODUCTORY STATEMENT.

In collecting and disseminating information on methods of mining and preparing domestic manganese ores, careful consideration was given by the bureau to the important matter of costs. Unfortunately few operators kept systematic cost sheets, hence the data collected are incomplete and unsatisfactory, particularly for an industry having so little uniformity. Domestic manganese deposits, especially deposits of ferro-grade ores (35+ per cent metallic manganese), vary widely, not only in the character of the ore, but also in the conditions of occurrence. Costs must therefore be expected to vary correspondingly, even when cost-keeping methods are uniform. But when the methods are as variable as the costs, computation becomes still more difficult.

It has, nevertheless, been possible by carefully compiling the material at hand to arrive at certain estimated average figures which are believed to be approximately correct, and to draw therefrom certain general conclusions. It must be clearly understood, however, that these average figures may be entirely misleading if applied to some one particular deposit. The conditions are so variable that it is necessary to consider each deposit strictly on its individual merits when considering its commercial possibilities. The average figures, however, have distinct value as guides to the probable competition which will have to be faced.

In fact these average figures have been compiled with this latter point especially in view. For this reason the more immediate foreign competitors, Brazil and Cuba, are considered as well as the domestic sources.

The subject may be approached in two ways—by determining so far as possible the actual cost, and by determining what may be called the "proper" cost. It is thoroughly appreciated that at many properties, owing to a multitude of reasons, costs have been unduly and unnecessarily high. However, as regards the costs of producing manganese ores during the war, it would serve no useful purpose to estimate what the cost ought to have been. War costs have become a matter of history, but the actual figures as bearing on probable future costs are of vital importance. When it comes to estimating the future, the opposite angle of approach may be taken. Domestic ores will soon once more have to face active and unrestricted foreign competition, and "proper" costs will have to be attained in order to keep the industry alive. Furthermore, if the question of protection be raised, only "proper" costs should be considered. It is appropriate, therefore, to discuss future possibilities from the second point of view, namely, as regards what the costs should be.

Another vital question, in every discussion of this sort, is what constitutes cost. Strange as it may seem, the important factors of amortization of plant and depletion of ore reserves are more often overlooked than not. This has no doubt been especially true of a business like that of the manganese industry during the war, chiefly characterized by the hurried and ofttimes ill-considered undertakings of many small operators with little financial strength. Nevertheless, owing to the uncertain life of the business, the writing off of the investment should have been the prime consideration.

In most instances the cost data collected evidently do not include allowances for amortization and depletion. It is now obviously impossible to say what these items should have amounted to; therefore, where they do seem to have been included, they will be omitted from the following discussion, for uniformity. This fact, however, should be kept in mind, if any comparison of the average estimated war costs with the prices then prevailing is attempted. The apparent profit which such a comparison shows probably failed, in a large number of cases, to offset the investment before the business collapsed upon the signing of the armistice. In fact, were not this the case, there would be little ground to-day for any claims under the War Minerals Relief act.

The same considerations must be kept in mind as regards the estimated average future costs. These are not intended to include appropriate charges for writing off investment.

COSTS OF DOMESTIC MANGANESE ORES.

The domestic manganese ores of ferro grade fall into two distinct classes, which vary widely from one another in their cost of production. These are (1) the carbonate ores, and (2) the oxide ores. A third class, namely, the silicate (rhodonite) ores, should perhaps be mentioned, but such ores in the natural state are too siliceous, and concentration too costly, to make their use feasible even with a war-time market. Consequently the small amounts produced were incidental to the production of carbonate and oxide ores, and they need not be considered separately.

CARBONATE ORES.

Carbonate ores were mined in quantity in only one locality, namely, Butte, Mont., but the output for 1918 was more than 20 per cent of the total output of manganese ores mined in the United States. The great bulk of the ore required no concentrating and little if any hand sorting. Mining was of the underground type, but the deposits were persistent and the mining operations were efficient. Furthermore the haul to the railroad was short, and accomplished by motor trucks over well-paved roads. It is understood that the cost on board railroad cars was not more than $5 per long ton—probably less. As the product contained 36 to 37 per cent metallic manganese, the unit cost was about 14 cents.

A small amount of siliceous carbonate ore was produced in the Butte district and concentrated at a neighboring customs mill, where the milling cost is said to have been $4.50 per ton of product.

OXIDE ORES.

The oxide ores occur mixed with clay, chert, and other gangue materials and invariably require careful hand cleaning or mechanical washing or concentrating. The deposits are generally erratic. Production costs have therefore necessarily been much higher than with the carbonate ores. In a few places carbonate ores were encountered as the weathered oxide ores were followed downward, but these carbonate deposits were too small and generally too impure, to lessen the otherwise high costs. Occasionally silicate (rhodonite) ores were encountered with depth, when mining as a rule had to be abandoned.

The oxide ores are widely scattered over the United States. They represented nearly 80 per cent of the 1918 production. In estimating the average cost of producing them, they will be treated as a group, although individual costs as between States, districts, and even mines in the same district, frequently varied within wide limits.

These variations were principally due (*a*) to inherent differences in the deposits as regards their size and richness, and (*b*) to their situation as affecting the cost of hauling to the railroad. Costs may best be discussed therefore under two main heads, namely (*a*) cost at the mine, and (*b*) cost of transportation to the railroad.

COST AT THE MINE.

Ordinarily it would be found convenient to subdivide the cost at the mine into mining cost and treatment or preparation cost. A large part of the manganese ore produced, however, was hand cleaned by picking or screening, and the available cost data do not differentiate between mining and hand cleaning. Consequently, three kinds of costs must be considered under mining and preparation, namely, mining, milling, and mining plus hand cleaning.

Figures for these costs as given below include labor, supplies, and the usual overhead charges, such as royalty. It will be of interest to examine these items separately before taking them up as lumped together under mining and treatment.

LABOR.

The labor supply, never abundant in many of the manganese districts, was seriously affected by conditions resulting from the war, not the least of which was the draft. Table 14 following shows data regarding wages recently paid.

TABLE 14.—*Wages paid in several of the manganese-producing States.*

(Dollars per day.)

	Arizonia.	Arkansas.	Georgia.	Montana.	Nevada.	Tennesee.	Utah.	Virginia.	Washington.
Carpenter			3.25	6.00		3.00			4.00
Common labor	4.75	3.00	2.75	5.50	5.00	2.50	4.00	2.00	
Underground mines			3.75	5.50					5.00
Open-cut mines		3.00	2.75						4.50
Steam-shovel operator		5.75	5.00						
Steam-shovel helper		2.75	4.00						
Dinky driver			3.75						
Fireman		2.75	2.75						
Mill men		3.75	3.00	5.50		3.00			4.50
Hand picking			2.50						
Mine foreman		6.00		7.50		3.50			
Mill foreman		4.00		8.30					
Blacksmith			3.25	6.00		5.00			5.00

Sufficient data are not at hand for satisfactory comparison between prewar and war times, but a few figures in this direction are available. For example, in 1916 the wage paid common labor in the Cartersville district, Georgia, was $1.25 per 10-hour day; a year later it had risen to $1.75, while in 1918 it was $2.50 per 8-hour day. In certain localities a bonus was paid for a maximum number of days' work per week, which raised the wage to $3 per 10-hour day. A similar condition existed in the Batesville district, Arkansas, and undoubtedly such conditions were universal through the manganese fields of the country. A wage of $2.50 for eight hours is probably a fair average.

SUPPLIES.

No detailed information as to cost of supplies per ton of product is available. The item was far higher, possibly double, the prewar cost.

ROYALTY.

A royalty commonly demanded during the war was 10 per cent of the gross value of the product, that is to say, of the selling price. The royalty thereby varied automatically with the grade of the ore.

Occasionally an arbitrary sliding scale was used, of which the following is an example:

Example of arbitrary sliding scale of royalty for manganese ore.

Manganese content of ore, per cent.	Royalty per ton.
20 or less	$0.50
20 to 30	1.00
30 to 40	1.50
40 and over	2.00

This scale provided for the higher grades of manganiferous ores as well as for ferro-grade ores. At some places, where the ore ran fairly uniformly as to grade, a single fixed rate of royalty was paid, as for instance 50 cents per ton at the Cason mine, Batesville, Ark.; or $1.50 per ton in the Erickson district, Utah.

The average manganese content of all ferro-grade oxide ores mined in the United States was approximately 41 per cent, and the average royalty paid for such ores may be taken to be $1.50 per long ton. Before the war, the current royalty for rather better grade ore was $1.00.

MINING AND TREATMENT.

The mining cost per ton of product naturally varied widely, not only with the cost of handling the bank dirt, or crude ore, but also with the yield of concentrates. As a rule, however, cost and yield roughly offset each other. Thus, a lean bank dirt had to be mined more cheaply than a rich one. Yields varied from 2 or less up to as much as 10 tons of bank dirt per ton of concentrates; and mining costs, where the ore was mined as it came and was sent to a mill for treatment, were found to range from $5 to $10 per ton of product, with the average not far from $7.50.

Milling costs likewise varied with the grade of the bank dirt, as also with the degree of refinement to which the process was carried. Concentrating plants varied from simple log washers, with or without picking belts, to elaborate mills with screens, jigs, and even occasionally tables. Water was not infrequently a costly item. Costs per ton of product varied from $5 to $8.50, the average being approximately $6.50.

At many small operations the bank dirt was simply picked or screened by hand. A profit in such cases was only possible when the crude material was fairly rich, but frequently cheaper costs were attained than at the larger and more elaborate operations, chiefly because overhead charges were reduced to a minimum, and the "mine" was readily moved from point to point, following the richer and cheaper ore. The average cost per ton of product was $12 to $13; individual costs were frequently less.

Considering the respective proportions of ore hand cleaned and milled, the average cost of the total product is estimated to have been $13.50. The figures that contribute to this average include a certain amount of the overhead charge in many cases, but probably $1.50 should be added to cover royalty, sampling, and similar overhead charges, which in other cases have not already been included. Thus it is concluded that the total average cost at the mine of the oxide ores was approximately $15.

The following individual examples are given as illustrating some of the details underlying the above conclusion.

Mr. F. G. Moses of the bureau's staff reported mining costs per ton of product in the Erickson district of Utah as given below. Costs at the mine were comparatively low at that point.

Costs per ton of product in the Erickson district, Utah.

Mining	$5.00
Overhead	2.00
Sampling	1.00
Royalty	1.50
Total	9.50

The average cost of mining per ton of product in the Cartersville district, Georgia, was about as follows:

Average costs per ton of product in the Cartersville district, Georgia.

	Underground.	Open cut.
Mining	$3.00	$2.50
Timbering	1.50	......
Royalty	1.50	1.50
Handling and miscellaneous	2.25	.50
Total	$8.25	$4.50

The bulk of the ore was obtained from open-cut workings and the average cost per ton of product is assumed to be $5.50. This is by far the most important manganese producing district in the State of Georgia.

The following rates paid for contract mining in the Batesville (Arkansas) and Cartersville (Georgia) districts are interesting.

Rates for contract mining per ton in Batesville and Cartersville districts.

Ore.	Batesville.[a]	Cartersville.[b]
Low grade	$8	$17
Medium grade	$12 to $15	....
High grade	$25 to $40	....

A high price was paid in Batesville for high-grade ores, yet the amount of ore that could be mined was much less than with low-grade ores, so that the wage earned was about the same.

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[a] The wash dirt mined to be cleaned at company's expense.

[b] For all lump and cleaned ore.

The following figures are given as approximate milling costs per ton of concentrates at a large operation in Virginia:

1917: 729 tons were cleaned at a cost of $10.77 per ton.
1918: 945 tons were cleaned at a cost of $6.53 per ton.
1918: Estimated cost for remodeled mill, $6.50 per ton.

The total estimated cost per ton of product on board railroad cars, including all charges, was given for the latter part of 1918 as $15. This is only one of a number of relatively important operations in Virginia.

Theodore Simons reported the following average costs for July, 1918, in the Philipsburg district, Montana. These costs are per ton of product on board railroad cars and include all charges for labor, timbering, hand cleaning or concentrating, road construction, development, repairs, and overhead:

Average costs in Philipsburg district.

Mining Company A	$15.14
Mining Company B	21.00
Mining Company C	16.00
Mining Company D	17.33
Mining Company E	17.00
Mining Company F	18.00

The arithmetical average of the above is $17.41 per ton of the product. As company A was by far the largest single producer its lower cost would bear down this figure in a weighted average to approximately $16.50, or roughly $15, excluding the cost of hauling and loading into railroad cars.

A representative milling cost at Philipsburg was reported as being $7.14 per ton of product. Custom mill work on Butte and Philipsburg oxide ores is said to have cost $2.50 per ton of crude ore. As the ratio of crude to concentrates was approximately 2.6 to 1, this latter figure would become about $6.50 per ton of concentrates.

COST OF TRANSPORTATION TO RAILROAD.

The location of the mines worked for manganese during the war with reference to the nearest railroad shipping point varied as widely as the other factors entering into cost. Not infrequently the distance hauled differed greatly even within the same district. For instance, in the Batesville district, Arkansas, hauls were commonly 2 to 3 miles, but hauls of 5 to 10 miles were not uncommon. The ore from a mine in Tennessee was hauled 14 miles to the railroad, and in the Buena Vista Valley, Nev., the distance hauled was 20 miles.

Transportation was by wagons and motor trucks. The average load was 1 ton. This average figure frequently applied even when 2-ton trucks were used, on account of poor roads. A few 5-ton trucks were used, but the tonnage handled in such units was small.

Table 15 following gives cost data and other data relating to a number of typical and important camps. The average cost of hauling for all oxide ores was roughly $3.50 per ton. To this should be added an average charge of 35 cents per ton for handling the ore at the two terminals of the haul.

TABLE 15.—*Cost of handling and hauling ore in manganese-producing States.*

State.	Average distance hauled, miles.	Load hauled, pounds.	Method employed.	Cost per ton.			
				Loading cars.	Hauling to railroad.	Loading wagons.	Weighing ore.
Arkansas	3½	3,300	Wagons and motor trucks	$0.25	$2.60		$0.10
Georgia	3.5	2,800	Wagons		1.10		
Montana	[a] 1½	3,000	Wagons and motor trucks	[b] .29	1.45		
Nevada	10	4,000	.....do		5.95		
New Mexico	18	2,800	Wagons	.45	3.50	$0.35	
Tennessee	2½	3,000	.....do		2.50		
Utah	13	2,800	.....do	.50	6.50	.50	
Virginia	2½	4,000	.....do	1.00	1.50		

[a] Philipsburg district only. [b] Cost of loading both wagons and cars.

SUMMARY OF COST OF OXIDE ORES ON BOARD CARS.

Adding the cost per ton of product at the mine, $15, to the cost for transportation to railroad plus handling, $3.85, the total estimated average cost on board railroad cars for the oxide ores is seen to be $18.85, or, say, $19 per long ton. With 41 per cent average metallic manganese content the unit cost is 46.4 cents.

COST OF RAILROAD TRANSPORTATION TO MARKET.

Railroad freight rates advanced materially during the war. Certain reductions were later made in favor of western shippers of manganese ores, but were still high. Some typical rates are given below.

Freight rates between Southern points and Chicago.

From—	Rate per ton of 2,240 pounds.
Northeastern and northwestern Georgia	$6.90
Eastern Tennessee and western North Carolina	5.90
Virginia	6.30
Abbeville and Greenville, S. C., and Lincolnton, Ga	8.40
Batesville, Ark	4.60

Freight rates between western points and Chicago, Philadelphia, and Birmingham.

State (any point).	Destination.	Rate per short ton.	Destination.	Rate per short ton.
Colorado	Chicago, Ill	$7.00	Philadelphia, Pa	$8.50
Oregon	.....do	11.00	Birmingham, Ala	12.50
Washington	.....do	11.00	.....do	12.50
California	.....do	11.00	.....do	12.50
Montana	.....do	8.00	.....do	10.50
Arizona	.....do	9.00	.....do	9.00
Nevada	.....do	10.00	.....do	11.00
Utah	.....do	9.00	.....do	10.00
New Mexico	.....do	7.00	.....do	7.00

The Government schedule of prices on which ores were sold during the war fixed the price per unit delivered at Chicago. The producer calculated the value of his ore at Chicago and then deducted the freight rate from point of production to Chicago to get the selling price of his ore f. o. b. railroad cars at shipping point. The buyer paid the actual freight from shipping point to point of consumption.

Whether ores are sold in the future f. o. b. cars at the shipping point or delivered at the furnace makes no difference to the producer. It is at the furnace that he must compete with other producers, whether domestic or foreign, and he must therefore compute the freight as part of his cost of production.

A small amount of ferro was made in western electric furnaces during the war, but the great bulk of it was made east of the Mississippi, at Chicago and in Alabama, Virginia, Pennsylvania, and New York. The production of Pennsylvania furnaces largely predominated, and this will unquestionably be even more true in the future, as production outside of Pennsylvania was principally stimulated by war prices.

It has already been pointed out that costs at the mine and costs of transportation to the railroad varied widely from point to point. These variations were not due to the section of the country in which the mines were situated, however, but rather to the nature of the deposits themselves. Two distinct groups of deposits have been differentiated, namely, the carbonates and the oxides. Within these groups it mattered little whether the mine was situated, for example, in Virginia or in California; the cost might be high or low in either place. Therefore it was found desirable to average all costs for each group and so determine an estimated average cost for producing ore of that group, irrespective of its locality.

It seems necessary to consider the matter of railroad freights from another angle. If all oxide ores, for instance, had been produced in Virginia, their average cost delivered at Pennsylvania furnaces would have been less than if a part had been shipped from points west of the Mississippi. Therefore, the freight for each State or district to the principal center should be weighted in an average according to the amount of ore shipped from that State or district. It has been necessary to make certain assumptions, as exact freight rates are in many cases not at hand, but the resulting figures without doubt approximate the truth.

Proceeding by this method and assuming that the principal consuming center is Pennsylvania, it is found that the average freight rate for the oxide ores during the latter part of 1918 was approximately $10.75 per long ton and for carbonate ores approximately $11.25, including war tax.

SUMMARY OF COST OF DOMESTIC ORES.

The foregoing estimates are summarized as follows:

Summarized data on estimated average cost of domestic ores.

	Carbonate ore.	Oxide ore.	Average.
Estimated tons produced in 1918	63,000	231,500	294,500
Estimated cost on railroad cars	$5.00	$19.00	$16.00
Estimated railway freight	$11.35	$10.70	$10.85
Total estimated cost delivered	$16.35	$29.70	$26.80
Manganese, per cent	36.5	41.0	40.0
Estimated cost per unit, cents	45.0	70.7	65.0

Of course, the more favored districts could produce oxide ores more cheaply than the average figures given. Virginia, for instance, probably delivered ore to the furnaces at a cost of about 55 cents per unit. Cartersville (Ga.) and Batesville (Ark.) ores should have reached Birmingham, Chicago, or Pennsylvania furnaces for 55 to 60 cents per unit. Nearly 85 per cent of the oxide ores, however, came from the far western States and were subject to high railway freights. The carbonate ores paid a high railway freight but other costs were so low that these were probably the cheapest ores produced either on a ton or unit basis.

DISCUSSION OF FUTURE POSSIBILITIES.

The average costs computed in the foregoing pages are for war times and more particularly for the last six months before the signing of the armistice.

In estimating future costs we have little to guide us in the way of prewar costs. Then only small amounts of domestic ores were mined, frequently in a small way by farmers when not busy with farm work. The production was in the southeastern States, where freight rates to consuming centers were not excessive. Probably only small profits were made, but these were satisfactory under the circumstances. Costs were $4 to $8 per ton on board railroad cars, but these costs are not fair guides to future possibilities.

Aside from railway freights, the wage scale enters more largely into the cost of production than any other single item; and it is probable that reduction in cost will come chiefly from reduction in wages. Better practice should account for some reduction, and cost of supplies may perhaps be properly expected to drop about in proportion to reduction in wages.

The consideration of future wages is a far-reaching problem, involving infinitely more important national industries than manganese mining, and an infinite number of correlated problems. The subject is ventured upon only because it seems desirable at this time to attempt some reasonable forecast of the probable future cost of manganese.

There is admittedly much difference of opinion in this respect, but it may fairly be assumed that wages must inevitably drop. The cost of labor is now approximately double the prewar figure in many of the more important manganese-producing districts. An instance has already been given where it has more than doubled. Its advance has frequently outsped the increase in cost of living, which is said in general not to have increased more than approximately 50 per cent in most manganese mining camps. The economic pressure of competition must either depress these wages or else the industry must be abandoned. If it be assumed that wages will drop, to fall in line with the increased cost of living, and that the latter will not decrease, then the drop in wages will be equivalent to 25 per cent of the war wage.

In this connection it may be further assumed that the cost of supplies will decrease in like proportion; whereas better practices will prevail and more efficient labor will be available than under war conditions. The net result is an estimated future reduction in cost of manganese ore on board railroad cars of 25 per cent.

It does not seem likely that railway freights will be materially lowered in the near future, except perhaps to the extent of the removal of the war tax.

On the basis of the foregoing premises, the summarized costs for war times given on page 101 become as follows:

Revised estimated average cost data on domestic manganese ores.

	Carbonate ore.	Oxide ore.	Average.
Estimated proportionate tonnage, per cent	21	79	100
Estimated cost on railroad cars	$3.75	14.25	11.72
Estimated railway freight	$11.00	10.40	10.53
Total estimated cost, delivered	$14.75	24.65	22.25
Manganese, per cent	36.5	41.0	40.0
Estimated cost per unit, cents	40.4	60.0	55.6

Here again, in certain more favored districts, as regards freight rates, the oxide ores can undoubtedly be delivered to the furnaces for less than 60 cents per unit. For the Southern States, generally, rates of 50 cents per unit or less should be attained, whereas for Western ores the cost would probably exceed 60 cents. The Butte carbonate ores are the one exception in the far west, owing to exceptionally cheap mining and hauling costs.

COSTS OF FOREIGN ORES.

The principal source of foreign ore during the war was Brazil. The next most important single source was Cuba. Smaller amounts came from other points in the West Indies, Central and South America, and the Orient, chiefly India.

In prewar times, Russia and India were more important sources than Brazil. It seems probable, however, that Russia will not for a few years become a large factor once more, and that the Indian output may also be slow in reaching this country; though English ferromanganese made from Indian ores is likely to appear before long in quantities sufficient to affect the market.

Our more immediate concern is with Brazilian and Cuban ores. Some information regarding costs in these countries is reviewed in the following pages.

CUBA.

Manganese mining in Cuba is generally by open-cut methods. In only one instance has there been anything approaching true underground work. Preparation for market is generally by hand cleaning, though log washers are used at the most important mine in the island. One concentrator, involving screens and jigs, was built and operated during the war.

The price of labor rose rapidly during 1917–18, being at the last as much as $1.80 to $2 per 9-hour day.

Mining and treatment costs naturally varied from point to point, ranging between $5 and $8 per ton of product.

Much of the ore was hauled from mine to railroad by mules or oxen, the cost varying, with the distance, from $2 to $10 per ton. A small amount was even packed on mules at a cost of $10 and upward.

Freight rates to shipping port ranged from less than $1 a ton up to $2.50 and more, according to the length of haul. Charges for loading into ship were sometimes as high as $1.50. Lighters were used in Santiago harbor.

Royalties were advanced materially during the war period. Before the war, $1 per ton was usual, and some leases were made for 80 cents. The figures for 1918 ranged from $1.50 to $2.50. In one instance a sliding scale was used, based on the delivered value of the ore. This scale started at $1,75 for ore worth $35 per ton, and advanced 25 cents for every $5 additional value for the ore. This meant approximately $2 for 40 per cent ore.

The foregoing figures are assembled for convenient comparison in the following table:

Manganese ore costs in Cuba.

Mining and treatment	$5.00 to $8.00
Hauling to railroad	2.00 to 12.00
Cuban railroad freight	1.00 to 2,50
Port costs and charges	0.75 to 1.50
Royalty	1.50 to 2.50

The two largest producers in Cuba were situated, as regards the bulk of their output, directly on the railroad or on tram lines leading to it. This factor must be considered with others in estimating the average cost. The total cost of ore in the ship is taken as $14 per ton, including the above items and some further miscellaneous and general charges. It is, of course, understood that no amortization of investment is included in this figure.

The actual costs at a small operation whose daily output was 12 to 20 tons of hand-cleaned shipping products were as follows:

Detailed costs for one Cuban operation.

Mining and cleaning	$6.50
Hauling to railroad	2.05
Surface rights	.80
Miscellaneous and general	2.00
Railroad freight to Cuban port	2.02
Handling into ship	1.50
Total per ton of product	14.87

The operators owned the ore deposit and therefore paid no royalty. They were heavily penalized, however, by the owners of the surface.

The 1916 costs at another operation, situated on the railroad and at which the ore was washed in log washers, were as follows:

Detailed cost at another Cuban operation, 1916.

Mining and washing	$5.50
Miscellaneous and general	1.00
Surface rights and royalty	1.10
Railroad freight and switching	1.79
Handling into ship	.72
Total	10.11

The costs of this latter operation were probably $3 to $5 higher during 1918.

BRAZIL.

Below is a statement of costs of mining manganese ore at four operations in Brazil.

Cost data from four operations in Brazil.

Item.	Operation.			
	A.	B.	C.	D.
Mining, f. o. b. cars	$5.00	$4.00	$5.00	$5.00
Export tax	1.85	1.85	1.90	1.85
Railway freight rate to Rio de Janeiro	1.50	1.50	[a] 5.35	[a] 5.40
Dockage at Rio de Janeiro	.65	.65	....	....
Lighterage at Rio de Janeiro	....	....	2.50	2.50
Transfer from narrow to broad gage railroad	....	....	.25	.25
Haulage to station	....	....	....	2.50
	9.00	8.00	15.00	17.50

[a] Companies C and D are handicapped by increased freight rates. The other two companies are still operating under old contracts.

In addition to the above figures a States tax of $3.20 was imposed during the war. Further, royalty charges up to $1.75 per ton were not uncommon. The total range of costs for Brazilian ore into ship would therefore be $12 to $22.50 per ton. As the low cost is for one of the largest and most important mines, it is probably that a weighted average would be approximately $15.

OCEAN FREIGHT RATES.

Freight rates on manganese ores from Brazil, Cuba, and India to the United States and England are given below:

Freight rates on manganese ores by vessel to United States and England.

Pre-war,	India to the United States	$4.32
Do.	India to England	3.84
Do.	Brazil to the United States or England	2.88
Do.	Cuba to the United States	1.50 to 2.50
1916,	Brazil to the United States	5.50 to 6.50
1918,	Brazil to the United States	15.00
1918,	Cuba to the United States	9.50

The rates of $9.50 and $15 per long ton of manganese ore may be taken as representative, respectively, for Brazil and Cuba during 1918. These figures are just about five times the pre-war rates. It is understood that they include insurance.

The average rail freight during the war from Atlantic seaboard to consumer was probably about $3 per long ton.

SUMMARY OF COST OF FOREIGN ORES.

Assembling the foregoing figures, we have a total estimated average cost for ore delivered at the furnace in 1918 of $23.50 for Cuba and $30 for Brazil. On a unit basis this would be 62 cents for Cuba (for ore containing 38 per cent manganese) and 67 cents for Brazil (for ore containing 45 per cent manganese).

DISCUSSION OF FUTURE POSSIBILITIES.

Before the war such ore as was shipped from Cuba was of a higher grade than that mined during the war, and was sold at the Atlantic seaboard for $10 to $12 per ton, or for, say, $8 to $10 per ton into ship at a Cuban port. This was very close to the average cost at that time. It is not likely that the cost of ore from Cuba will drop in the future much below $11 to $12, but it is certain that ocean freights will drop materially, not improbably to the pre-war figure.

On this assumption we may expect to see Cuban ores reach the United States Atlantic seaboard in the not remote future at a cost of about $13.50, or, say, $16.50 a ton delivered at the furnace if the United States rail freight is not reduced. The quality of the ore, in order to be saleable, will have to be better than it was during 1918. If it contains an average of, say, 41 units of manganese, the estimated future cost delivered to the furnace becomes 40 cents per unit.

The Brazilian problem is somewhat more complicated, as it is uncertain what will be done by the Brazilian Federal and State

Governments with regard to the present high taxes. Other Brazilian costs will probably not be much reduced. If we assume that the taxes, which in 1918 amounted to practically $5 per ton, are reduced two-thirds, the estimated average cost of $15 becomes $11.67, or say $11.50 to include some possible slight reduction in other directions.

It is interesting to compare this estimate with the selling price of Brazilian ores at Rio de Janeiro in former years, which were as follows:

Selling prices of Brazilian ores at Rio de Janeiro, 1914–1918.

Year.	Selling price per ton, ries.	Approximate equivalent.
1914	22, $000	$7.50
1915	29, $000	10.00
1916	55, $000	13.50
1917	93, $000	23.25
1918	117, $000	29.25

The real difference in the future will come in the matter of ocean freights. If these drop to nearly their former level, it should be possible to deliver Brazilian ores to the United States consumer for approximately $18.50 per ton. This figure includes $13.50 for cost into ship at Rio de Janeiro, $3 ocean freight, $1, say, for insurance, and $3 for United States rail freight to the furnace.

Brazilian ores reaching this country were formerly of higher grade than those imported during the war. If the grade now improves slightly, say to 46 per cent manganese, the estimated average future unit cost becomes 40 cents.

Thus it may fairly be expected that both Brazilian and Cuban ores can before long be delivered to the United States consumer for about 40 cents per unit. At an equal price, Brazilian ore will always have the preference owing to the higher content of metallic manganese.

COMPARISON OF DOMESTIC AND FOREIGN COSTS.

The estimated average costs for manganese ores, per long ton delivered to the consumer, are compared in the following table:

Estimated average costs of manganese ores delivered to the consumer during 1918 and in the future.

Kind of ore.	Estimated average cost during 1918.			Estimated average cost in the future.		
	Cost per long ton.	Average manganese content, per cent.	Cost per unit, cents.	Cost per long ton.	Average manganese content, per cent.	Cost per unit, cents.
Domestic:						
Carbonates	$16.35	36.5	45.0	$14.75	36.5	40.4
Oxides	29.70	41.0	70.7	24.65	41.0	60.0
Foreign:						
Cuba	23.50	38.0	62.0	16.50	41.0	40.0
Brazil	30.00	45.0	67.0	18.50	46.0	40.0

This comparison indicates that the oxide ores could not compete with foreign ores even when protected by high ocean freights. Had foreign ores moved freely and been sold at strictly competitive prices, the domestic manganese industry could not have developed except perhaps in certain more favored districts in the Southern States, and in the Butte district (carbonate ores). The urgent need for diverting shipping into other channels, with the consequent restrictions and embargoes on imports, together with the fixing of high prices in order to stimulate production to replace such imports, created highly artificial conditions which practically removed the element of competition. In short, the ores had to be had "at any price."

The comparison further shows that when ocean freight rates return to more nearly their normal level, even the most favored oxide ores, as well as the carbonate ores, will have hard going in a competitive field.

Looking into the more remote future, it is not impossible that the costs of Cuban and Brazilian ores will have to be reduced still further in order to compete with Indian and Russian ores, thereby placing domestic ores at a still greater disadvantage unless domestic costs are correspondingly decreased.

It must furthermore be remembered that at the same price per unit the higher grade ore will always have the preference. This fact is reflected in the invariable sliding scale of prices, the higher price per unit being paid for the higher grade ore. In smelting manganese ore, not only does it cost money to melt and remove the impurities as slag, but the loss of manganese is more or less directly proportionate to the volume of the slag. Domestic ores, being generally of lower grade than foreign ores, command a lower price.

Other factors which militate against domestic ores in competition with foreign ores are that they are variable in grade while foreign ores are far more uniform, and their production is irregular while the foreign supplies are generally dependable.

CHAPTER 8.—PRODUCTION OF MANGANESE ALLOYS IN THE BLAST FURNACE.

By P. H. ROYSTER.[1]

INTRODUCTORY STATEMENT.

This chapter gives data collected by the Bureau of Mines on the operation of blast-furnace plants in the United States producing manganese alloys. This information should be of value to the furnace operator; certainly, without necessarily agreeing with any generalizations and conclusions introduced, he can determine from the data assembled in Table 17 what has been done by others with raw materials and furnace conditions approximating his own, provided comparable combinations can be found in the results presented.

The operation of a blast furnace even for making pig iron is highly empirical. Past furnace records and previous experience are nearly as necessary to successful operation as good coke and a hot blast. As in the past, the production of manganese alloys in this country has been comparatively small, and most of the men in charge of manganese furnaces have had relatively little experience on such ores. Hence, the information obtainable from the few records extant is presented herein in considerable detail, to compensate in part for the paucity of data that can be consulted by furnace operators.

The attempt has been made to have this investigation cover the widest possible range of materials and operating conditions. Data from 18 furnaces will be found in the tables following. The information was obtained in the summer of 1918, and includes every furnace in blast that was making manganese alloys, with the exception of the furnaces of the Carnegie Steel Co., and of the Lavino Co., which were unwilling to cooperate in the investigation. The companies which by their generous cooperation have made the report possible are the American Manganese Co., Bethlehem Steel Co., B. & B. Trading Co., Buffalo Union Furnace Co., Colorado Fuel and Iron Co., John B. Guernsey & Co., Donner Steel Co., E. E. Marshall, Miami Metals Co., New Jersey Zinc Co., Seaboard Steel & Manganese Corporation, Southeastern Iron Corporation, and Wharton Steel Co.

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FURNACES INVESTIGATED.

The 18 furnaces investigated have been assigned letters and are designated throughout this report by these letters. The physical dimensions of 14 of these furnaces are given in Table 16 following.

TABLE 16.—*Dimensions of 14 of the 18 blast furnaces investigated.*

[All dimensions in feet.]

Furnace.	Distance from stock line to tuyères.	Bosh diameter.	Hearth diameter.	Stock-line diameter.	Distance from tuyères to hearth bottom.	Bosh height.
A	62.1	13.4	8.8	(?)	5.9	8.8
B	55.0	16.5	9.5	(?)	(?)	11.2
C	66.2	14.3	9.5	10.0	5.0	9.3
D	55.8	14.8	9.5	10.0	8.3	17.7
E	63.5	16.0	10.0	10.0	6.5	(?)
F	57.8	15.2	10.5	12.5	7.6	12.5
G	63.0	18.0	11.5	13.5	8.3	12.1
H	53.6	17.0	12.5	13.0	5.7	12.8
I	55.3	18.5	12.5	12.5	7.6	12.0
J	67.4	18.3	13.0	12.5	15.9	12.5
K	63.0	18.6	13.2	14.5	6.0	14.0
O	(?)	17.7	11.0	13.0	(?)	11.9
Q	65.5	17.5	12.5	12.9	6.3	12.5
R	66.0	19.0	13.0	13.0	6.0	12.5

COLLECTION OF DATA.

CHARACTER.

On every visit of the Bureau of Mines field party to the different furnaces the companies have opened for inspection all the furnace records, charge sheets, and chemical analyses in their possession. Periods of ten days' continuous operation were selected during which the various factors of blast temperature, silicon in the alloy, manganese in the slag, etc., were relatively constant, and all the figures applicable to such a period were copied and averaged. The average figures for such 10-day periods constituted the data on a single "experiment" or "run." The guiding principle in the selection of the experimental periods was to cover the extreme range of furnace conditions; for example, operation with high and low grade ores, and with both high and low blast temperature.

COMPLETENESS.

The information obtainable without undue trouble from a blast furnace includes the chemical analyses, the physical condition, and the weights of the ores, stone, coke, coal, scrap, and wind charged and of slag, metal, dust, gas and scrap produced, with the temperature observable in the hearth, the temperature of the blast entering the furnace, and the temperature of the metal, slag, and gas coming out of the furnace. Even with the aid of such data, it is seriously to be doubted whether anything like a complete description, of what takes place in a furnace can be given. In the present investigation, the data obtained fell short of what must be considered essential to a complete understanding of the smelting problem. Information regarding gas analyses, moisture in the blast, the amount and the analysis of the dust blown from the furnace was not obtainable in any case. Analyses of coke, by far the most important single analysis in the whole group, were rather fragmentary. Furthermore, the custom of limiting the slag analysis to a determination of silica, alumina, and manganese is widespread. Naturally no measurements of metal, slag, or tuyère temperatures had been made by the companies; nor had any gas analyses been taken, save in one case. The bureau's field party made a limited number of such temperature measurements at six furnaces and determined also the CO and CO_2 contents of the gas at five furnaces. On the whole, however, the information was far from complete.

ACCURACY.

Aside from purely analytical errors, the applicability of any given analysis is questionable for two reasons: (*a*) The sample analyzed may not have been representative of the pile or car from which it was taken; (*b*) the furnace records failed to give any connecting link between car and pile analysis and the material charged on any given day. Coke was charged by volume and not by weight, making difficult an estimation of the weight of carbon charged. It is obvious that greater accuracy would be obtained from experimental periods taken over a whole month or, better, six months. It must be remembered, however, that differences in operating conditions were being sought and long-period averages flatten out most of the variations from which it was hoped to obtain information.

In a number of runs, it was possible to obtain samples of slag, metal, stone, and ore, which were reanalyzed by the bureau's chemists. No detailed comparisons are given here but the combined effect of all the differences in the analyses introduces in the manganese balance an average uncertainty of about 15 per cent. Even if, therefore, the blast furnace were a precise and sensitive heat engine with a number of definite relations existing between its operating quantities, it is not to be hoped that these relations could be determined from the kind and quantity of data collected here.

TABULATED OPERATING DATA.

Data on ferromanganese production were obtained from 11 furnaces, and are presented in Table 17 in the form of 40 10-day periods. Similiar data from 10 furnaces making speigeleisen are presented in Table 18, in the form of 30 10-day periods. The data in these tables are classified under 23 items as applied to each of the 70 periods.

Items 1, 2, 2a, and 3 give figures on the ore, coke, coal, and stone charged, the units being in pounds per ton of 2, 240 pounds of alloy made. Items 4, 5, and 6 give a partial analysis of the ore mixture. Item 7 shows the tons of alloy made in 24 hours (not including scrap made). Item 8 gives the slag weight as calculated by the bureau and is a rather uncertain quantity. Two methods of calculating the weight of the slag, both of which should give the same answer, are as follows:

Method 1.

If a=per cent base in stone,
b=weight of stone,
c=per cent base in slag;
assuming there is no appreciable base in the coke or ore, then

$$\text{Slag weight}=\frac{ab}{c} \qquad (1)$$

Method 2.

If d=per cent silica in ore
e=weight of ore
f=per cent silica in stone
g=weight of stone
h=per cent silica in coke
i=weight of coke
j=0 .478 × per cent silicon in metal
k=per cent silica in slag, then

$$\text{Slag weight}=\frac{de+fg+hi-j}{k} \qquad (2)$$

On account of the various errors and omissions in the weights and analyses involved, it is unusual to find any close agreement between the results of the two equations; in some computations the slag weight may be given as 2,500 pounds by equation 1 and as 3,500 pounds by equation 2. Each value under item 8 is the average of the two answers resulting from equations 1 and 2.

Metal analyses for Mn and Si are given by items 9 and 10. The largest disagreement in checking manganese in the metal was 10 per cent, and in checking Si, 1.1 per cent. Values for the temperature of the blast and of the top gas appear in items 11 and 12. For a given furnace, these have some significance. A comparison of figures for different furnaces should be made cautiously, however. At some furnaces, the pyrometer will indicate a blast temperature of 1650° F. with a black blowpipe; at others the pyrometer will show only 1350° F. with a red-hot blowpipe.

Still less importance should be attached to the wind blown per minute as given by engine displacement in item 13. In period 1 on furnace A 12 pounds of carbon per minute was charged and 12,870 cubic feet of air per minute indicated. This is 105 cubic feet of air per pound of carbon. Even with no carbon blown out as dust, none taken up by the metal, and none absorbed in the stack, this means a tuyère combustion producing 35 per cent CO_2, which is impossible.

Items 14 to 18 show the slag analysis. In a number of calculations the lime and magnesia had to be computed together by difference. Item 19 gives the fixed carbon in the coke and coal charged per ton of metal. Items 20 to 23 give the manganese balance. Item 23 shows the percentage of the manganese not accounted for by that in the metal and in the slag. This figure includes practically every error in weights, analyses, and computations in the preceding 22 items and is undoubtedly the least satisfactory figure in the table:

TABLE 17.—*General data from operation of ferromanganese blast furnaces.*

Period No. Furnace	1 A	2 A	3 A	4 A	5 A	6 A	7 A	8 A	9 A	10 B	11 B	12 B	13 B
1 Ore.....pounds per ton..	5,981	5,303	5,870	6,000	5,090	5,475	5,720	6,166	6,990	7,560	6,212	7,545	6,025
2 Coke.....do....	4,658	5,475	6,290	7,133	5,016	5,628	5,595	7,352	7,836	8,070	7,776	8,435	7,534
3 Stone.....do....	1,980	1,780	1,826	2,320	1,632	1,806	1,860	2,172	2,112	2,700	2,180	2,475	2,265
Ore analysis:													
4 Mn.....per cent..	41.18	39.11	39.43	42.01	42.25	39.80	38.45	36.66	37.14	35.75	36.70	36.00	38.00
5 Fe.....do....	4.54	6.65	7.42	3.33	6.07	11.28	14.95	4.60	3.69	6.35	4.41	6.14	3.43
6 SiO_2.....do....	7.78	5.64	6.33	8.45	6.73	4.34	2.58	6.14	6.65	15.06	17.05	14.97	15.28
7 Ferromanganese.....tons per day..	45.7	37.2	29.0	33.3	35.7	40.9	41.5	29.5	31.2	33.8	35.0	33.8	34.7
8 Slag.....pounds per ton..	2,671	2,445	2,805	3,035	2,248	2,180	2,070	3,335	3,460	4,630	4,110	4,460	3,670
9 Mn in alloy.....per cent..	78.2	70.6	71.3	80.90	73.5	69.4	68.3	78.3	80.35	72.34	77.13	72.88	77.7
10 Si in alloy.....do....	.76	1.47	.92	1.24	1.24	.73	.85	.70	1.86	.62	.58	.74	1.43
11 Blast temperature.....°F..	1,308	1,268	1,196	1,232	1,432	1,279	1,303	1,175	1,150	786	770	886	782
12 Top temperature.....°F..	935	1,025	1,049	1,064	998	1,147	1,103	1,014	980	....	....	....	....
13 Air per minute.....cubic feet..	12,870	12,820	....	....	12,510	....	....	....	....	....	....	....	....
Slag analysis:													
14 CaO.....per cent..	27.2	24.1	23.9	30.1	26.5	30.1	33.1	38.2	50.2	39.5	44.6	39.1	44.3
15 MgO.....do....	9.7	8.9	8.6	10.3	9.6	10.9	11.5	11.8					
16 Al_2O_3.....do....	13.0	16.2	16.2	13.2	13.5	17.0	15.4	14.1	15.7	11.1	12.5	11.0	14.1
17 SiO_2.....do....	27.8	28.2	26.3	29.8	28.7	27.9	28.3	25.5	26.2	29.9	28.7	30.5	29.8
18 Mn.....do....	14.6	15.6	18.9	11.7	14.9	9.4	9.4	6.9	5.3	13.6	9.5	13.3	9.1
19 Carbon.....pounds..	3,810	4,440	5,360	5,820	4,060	4,560	4,480	5,992	6,380	7,090	6,810	7,400	6,600
20 Mn charged.....do....	2,462	2,075	2,316	2,520	2,149	2,180	2,200	2,260	2,600	2,700	2,280	2,715	2,292
21 Mn to alloy.....per cent..	71.2	76.4	69.0	72.0	76.3	71.2	71.0	77.8	66.8	60.0	75.8	60.1	76.0
22 Mn to slag.....do....	15.8	18.4	22.9	14.0	15.5	9.5	9.4	10.1	7.0	22.5	17.1	21.9	14.6
23 Mn lost in stack.....do....	12.9	5.2	8.1	14.0	8.0	19.3	19.6	12.0	23.7	17.5	7.1	18.0	9.4

Period No.	14	15	16	17	18	19	20	21	22	23	24	25	26
Furnace	B	F	F	F	G	G	I	I	J	J	J	J	K
1 Ore, pounds per ton	8,108	7,099	4,813	4,831	7,525	6,050	8,150	5,850	8,560	5,040	4,900	4,960	4,040
2 Coke, do.	9,510	7,539	6,092	6,431	5,460	4,863	7,282	4,652	6,940	500	4,900	4,955	6,275
2a Coal, do.	....	....	....	....	546	459	....	....	....	432	415	419	....
3 Stone, do.	3,320	3,168	2,091	2,084	2,941	2,675	3,160	2,165	2,516	1,216	126	1,255	1,985
Ore analysis:													
4 Mn, per cent	34.00	42.91	43.68	45.76	32.53	32.33	39.26	41.64	34.83	44.73	45.20	43.75	50.65
5 Fe, do.	7.75	4.09	6.71	5.90	5.60	5.60	2.40	5.54	4.00	4.06	3.80	4.18	7.27
6 SiO_2, do.	17.84	11.30	6.87	8.44	12.62	12.62	15.54	7.43	14.92	2.60	2.53	2.86	6.52
7 Ferromanganese, tons per day	35.0	32.0	44.9	44.7	56.5	60.7	50.0	82.7	60.8	94.1	70.0	97.9	46.9
8 Slag, pounds per ton	5,740	4.100	2,470	2,610	4,600	8,765	5,040?	2,555	6,230	2,050	2,020	1,985	2,190
9 Mn in alloy, per cent	74.5	76.3	79.1	78.8	78.2	67.5	71.4	77.1	69.1	79.2	79.7	79.3	75.9
10 Si in alloy, do.	.58	.37	.83	1.12	1.53	1.61	1.92	.56	2.05	0.75	0.63	0.61	2.43
11 Blast temperature, °F	830	735	993	1,027	1,140	1,183	952	1,277	1,108	1,156	1,257	1,172	950(?)
12 Top temperature, °F	....	757	328	143	730	780	1,016	934	849	496	....	562	....
13 Air per minute, cubic feet	....	14,380	22,500	22,500	17,500	16,900	17,100	16,500	20,000	15,900	20,000	19,500	....
Slag analysis:													
14 CaO, per cent	....	28.4	35.0	33.3	25.2	26.2	26.6	31.4	24.2	26.0	27.3	27.9	46.4
15 MgO, do.	35.5	6.5	9.9	11.2	10.7	12.8	14.1	12.6	11.1	8.9	7.1	7.1	1.7
16 Al_2O_3, do.	10.7	11.1	10.3	11.5	11.5	13.9	15.2	11.3	11.4	24.2	23.5	24.5	14.7
17 SiO_2, do.	31.8	29.1	27.5	27.8	29.8	30.9	27.7	26.0	30.4	21.7	21.3	20.9	27.3
18 Mn, do.	15.6	19.8	13.2	11.3	13.3	11.6	11.4	12.0	15.1	8.1	13.3	12.1	5.4
19 Carbon, pounds	8,340	6,220	5,030	5,310	5,140	4,563	5,990	3,825	5,550	4,395	4,340	4,434	5,420
20 Mn charged, do.	2,760	3,043	2,100	2,218	2,450	1,969	3,200	2,435	2,985	2,256	2,226	2,168	2,048
21 Mn to alloy, per cent	60.6	56.1	84.4	80.2	71.5	77.0	52.7	71.0	51.6	78.7	80.3	81.6	83.0
22 Mn to slag, do.	32.4	25.6	15.5	13.3	24.9	21.8	18.0	12.6	31.2	7.4	12.1	11.1	5.8
23 Mn lost in stack, do.	7.0	18.3	.1	6.5	3.6	1.2	29.3	16.4	17.2	13.9	7.6	7.3	11.2

TABLE 17.—*General data from operation of ferromanganese blast furnaces*—Continued.

Period No. Furnace	27 K	28 K	29 K	30 L	31 L	32 L	33 L	34 L	35 L	36 P	37 P	38 C	39 M	40 M	Average.
1. Ore.............pounds per ton..	4,480	4,650	5,215	6,770	6,565	5,312	6,222	5,465	5,200	7,000	4,117	5,245	6,586	6,598	5,992
2. Coke....................do....	5,802	6,368	6,329	7,458	8,459	6,500	6,421	4,855	5,164	5,350	5,965	5,704	7,118	7,100	6,320
2a. Coal....................do....	....	....	....	660	747	524	424	471	352	545	506	811	998	957	200
3. Stone....................do....	2,268	2,184	2,386	2,965	3,100	2,582	2,528	2,200	2,155	3,591	1,686	2,773	3,042	3,510	2,349
Ore analysis:															
4. Mn..................per cent..	51.13	50.11	46.94	39.97	39.68	35.45	35.38	32.38	33.53	38.70	45.50	40.70	45.11	46.36	40.33
5. Fe.....................do....	7.10	7.30	10.84	3.51	3.69	9.54	4.50	8.02	7.81	4.25	5.55	7.45	3.66	4.15	5.93
6. SiO_2....................do....	6.35	5.57	6.24	7.53	7.40	8.91	6.28	10.11	8.63	15.89	4.85	7.01	6.59	4.08	8.60
7. Ferromanganese....tons per day..	85.8	49.5	77.5	48.1	43.0	64.7	50.5	77.2	77.1	61.3	78.0	35.0	42.0	36.8	51.7
8. Slag............pounds per ton..	2,580	2,315	3,285	3,126	3,305	2,886	2,727	2,452	2,256	4,468	1,880	2,700	3,900	3,480	3,196
9. Mn in alloy............per cent..	74.8	75.9	62.5	80.7	81.1	69.8	78.4	70.4	71.8	78.1	80.9	74.9	75.3	74.5	74.9
10. Si in alloy................do....	0.29	1.82	0.26	1.29	1.47	0.75	2.72	1.46	2.22	....	....	2.19	1.20	1.30	1.15
11. Blast temperature...........°F..	970	950(?)	998	1,065	1,043	1,089	1,204	1,457	1,207	1,450(?)	1,450(?)	1,387	1,242	1,182	1,135
12. Top temperature...........°F..	965	....	845	1,162	1,152	1,162	1,127	1,123	1,140	750(?)	750(?)	52	905	910	690
13. Air per minute.......cubic feet..	17,500	....	19,100	17,800	17,700	18,800	17,800	18,800	18,800	....	....	....	....	....	....
Slag analysis:															
14. CaO................per cent...	37.1	46.6	33.3	40.4	47.7	42.0	43.3	41.9	44.3	41.0	45.6	29.8	38.8	43.8	41.75
15. MgO....................do....	1.1	1.5	1.4	6.8	5.7	5.6	4.3	6.3	4.9			18.2			
16. Al_2O_3....................do....	11.4	13.8	11.1	11.7	12.8	11.8	14.5	10.5	13.2	14.9	18.8	10.7	14.4	11.1	14.0
17. SiO_2....................do....	28.8	27.6	26.5	31.1	27.4	29.9	27.3	31.6	31.0	35.4	20.3	32.2	25.7	28.7	28.1
18. Mn....................do....	14.3	5.7	19.6	8.8	4.6	7.2	6.9	5.8	4.4	4.5	9.9	7.3	15.2	11.8	10.6
19. Carbon.................pounds..	4,690	5,450	5,280	6,740	7,655	5,951	5,650	4,432	4,568	5,060	4,700	5,280	6,500	6,471	5,323
20. Mn charged...............do....	2,288	2,330	2,442	2,710	2,608	1,888	2,200	1,770	1,745	2,718	2,060	2,138	2,970	3,056	2,382
21. Mn to alloy............per cent..	73.1	73.0	57.2	66.9	69.7	82.8	79.7	89.0	92.1	64.5	88.0	78.7	57.1	54.5	72.0
22. Mn to slag.................do....	16.2	5.7	26.3	10.3	5.9	10.1	11.1	8.1	8.7	6.4	4.9	8.9	20.0	13.4	14.7
23. Mn lost in stack...........do....	10.7	21.3	16.5	22.8	24.4	7.1	9.2	2.9	—0.8	29.1	7.1	12.3	22.9	32.1	12.8

TABLE 18.—*General data from operation of spiegeleisen blast furnaces.*

Period No.	41	42	43	44	45	46	47	48	49	50	51	52	53	54	55
Furnace	C	C	C	C	D	D	D	D	D	E	E	E	E	E	H
1. Ore....pounds per ton	5,192	5,795	5,488	5,365	4,510	4,505	4,516	4,420	4,282	5,930	6,560	6,945	6,550	6,800	4,525
2. Coke....do	3,327	3,831	2,923	3,150	3,110	3,180	3,060	3,185	3,200	3,600	4,590	4,640	4,518	4,100	2,970
2a. Coal....do	749	926	725	693	....	....	....	....	....	....	....	....	....	....	....
3. Stone....do	1,568	2,052	1,655	2,328	1,436	1,691	1,721	1,684	1,530	2,981	3,290	3,090	3,310	3,380	1,331
Ore analysis:															
4. Mn....per cent	12.72	12.77	12.77	12.77	14.62	13.62	14.13	13.84	14.06	13.35	12.55	11.27	12.29	12.91	14.85
5. Fe....do	36.72	36.90	37.65	37.65	40.20	39.90	37.83	39.33	41.46	31.87	30.84	34.71	32.27	32.18	40.68
6. SiO_2....do	17.46	16.65	17.41	17.41	9.78	10.20	10.22	10.55	9.09	19.80	20.81	21.13	22.30	20.20	9.62
7. Spiegeleisen....tons per day	44.7	30.2	46.0	51.0	88.3	95.0	95.8	94.7	74.9	83.0	73.0	64.0	66.0	69.0	113.3
8. Slag....pounds per ton	2,614	2,990	2,330	2,950	2,035	2,050	2,155	2,135	1,940	3,765	4,615	4,400	4,755	4,430	1,870
9. Mn in alloy....per cent	14.9	16.50	16.2	17.1	19.9	19.2	18.9	17.6	19.4	16.7	14.7	15.7	14.6	16.7	20.0
10. Si in alloy....do	3.1	4.9	2.1	1.2	2.3	2.6	1.0	1.3	1.3	0.7	0.8	2.4	0.8	1.1	0.9
11. Blast temperature....°F	1,234	1,304	1,344	1,211	1,178	1,166	1,132	1,194	1,100	983	914	1,068	914	1,005	1,208
12. Top temperatur.e....°F	613	749	540	538	700	600	500	400	393	....	....	593	....	....	....
13. Air per minute....cubic feet	8,660	8,740	8,660	8,800	....	....	....	....	15,660	....	....	....	....	....	20,600
Slag analysis:															
14. CaO....per cent	26.0	24.3	27.8	31.6	26.1	26.2	25.5	25.8	25.4	42.0	39.3	39.5	37.1	42.0	25.1
15. MgO....do	10.1	11.1	12.6	11.6	15.2	15.4	15.0	13.2	14.4						14.1
16. Al_2O_3....do	10.3	13.3	9.8	7.8	17.6	16.9	15.2	15.1	16.2	9.4	10.9	13.2	10.9	9.9	14.7
17. SiO_2....do	28.3	37.2	36.3	37.2	31.1	31.0	30.4	31.3	30.3	36.6	35.8	37.3	38.1	36.9	30.7
18. Mn....do	10.79	10.07	10.48	8.91	6.39	6.31	9.95	10.66	9.35	8.40	9.88	6.73	9.88	7.73	9.68
19. Carbon....pounds	3,292	3,740	2,870	3,045	2,820	2,880	2,780	2,890	2,910	3,000	3,830	3,870	3,760	34.20	2,710
20. Mn charged....do	660	740	695	681	660	614	638	612	601	793	823	780	802	879	672
21. Mn to alloy....per cent	49.0	50.6	53.4	55.1	67.5	70.0	66.1	64.3	72.2	47.2	40.0	46.4	40.7	42.1	66.6
22. Mn to slag....do	42.6	40.7	35.0	38.7	19.7	21.1	33.5	37.2	30.2	39.8	55.4	38.0	58.0	39.0	26.9
23. Mn lost in stack....do	8.4	8.7	11.6	6.2	12.8	8.9	0.4	−1.5	−2.4	13.0	4.6	15.6	1.8	18.9	6.5

Period No.	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	Average.
Furnace	H	H	J	J	M	M	M	M	P	B	Q	Q	R	R	R	
1. Ore, pounds per ton	5,100	4,300	5,710	5,000	5,025	4,625	5,145	5,665	4,431	5,322	6,050	5,852	5,495	5,818	5,355	5,520
2. Coke, do.	2,980	2,870	4,900	4,210	3,506	3,297	3,110	3,987	3,290	4,220	5,360	5,147	5,853	7,147	5,048	3,950
2a. Coal, do.	....	....	280	260	....	238	575	830	340	....	....	....	....	....	....	187
3. Stone, do.	1,423	1,290	2,155	1,980	2,104	2,026	1,882	1,762	1,450	1,830	3,491	3,921	3,728	3,946	3,480	2,324
Ore analysis:																
4. Mn, per cent	14.50	15.53	9.02	11.86	18.8	13.3	11.1	15.95	12.76	11.05	11.10	9.83	10.25	9.86	99.06	12.75
5. Fe, do.	37.67	40.92	31.60	31.84	31.8	32.3	35.1	32.14	....	....	32.43	32.37	32.35	31.24	31.24	35.10
6. SiO_2, do.	10.79	9.19	17.70	18.37	10.0	12.9	9.5	7.96	10.98	20.42	16.60	14.89	19.97	17.41	16.48	14.84
7. Spiegeleisen, tons per day	100.5	134.0	91.7	85.5	107.0	88.0	81.3	69.0	153.5	60.0	117.3	127.0	71.4	50.6	59.3	83.8
8. Slag, pounds per ton	2,150	1,572	3,450	2,830	2,600	2,500	2,100	2,450	1,810	3,960	4,835	4,060	4,585	4,600	4,735	3,109
9. Mn in alloy, per cent	18.7	20.2	16.7	19.9	17.0	19.2	18.0	16.1	19.3	17.8	17.8	17.0	15.9	16.7	16.1	17.48
10. Si in alloy, do.	1.0	2.3	2.8	2.5	0.7	0.5	10.8	91.6	....	1.9	0.8	0.9	2.9	2.5	4.4	1.77
11. Blast temperature, °F.	1,283	1,198	1,275	1,088	1,008	1,316	1,172	942	....	....	880	755	768	690	909	1,080
12. Top temperature, °F.	500	....	890	490	505	750	804	942	....	....	511	522	535	593	483	572
13. Air per minute, cubic feet	19,300	20,500	18,700	19,500	....	....	....	....	....	....	30,000	30,300	28,800	25,000	21,300	....
Slag analysis:																
14. CaO, per cent	23.9	27.1	30.2	27.7	40.2	49.7	53.5	39.8	43.7	46.4	49.0	50.8	44.0	46.7	46.1	42.18
15. MgO, do.	14.0	16.2	11.1	9.0												
16. Al_2O_3, do.	15.7	15.8	9.5	8.3	13.9	12.2	12.2	14.8	16.4	11.1	9.4	9.8	9.7	11.0	11.3	12.41
17. SiO_2, do.	30.8	30.9	30.8	33.9	34.8	30.9	31.2	30.2	30.5	34.5	35.1	34.1	36.5	35.4	36.3	33.48
18. Mn, do.	10.24	6.35	2.79	6.50	7.37	4.36	11.71	10.58	6.02	4.93	3.83	2.85	6.38	4.14	3.66	7.56
19. Carbon, pounds	2,720	2,620	4,398	3,780	2,800	2,796	2,864	3,735	2,870	3,630	4,370	4,195	4,790	5,820	4,110	3,444
20. Mn charged, do.	739	667	516	593	942	615	570	903	564	587	571	576	563	574	486	671
21. Mn to alloy, per cent	66.7	67.4	72.5	75.0	40.6	69.8	70.7	41.1	76.7	68.0	69.8	66.0	63.4	65.2	73.6	60.22
22. Mn to slag, do.	29.9	15.0	18.6	31.0	20.2	17.7	39.3	28.8	19.3	33.5	32.4	20.2	51.8	33.2	35.6	32.75
23. Mn lost in stack, do.	13.4	17.6	8.9	−6.0	39.2	12.5	−10.0	30.1	4.0	−1.5	−2.2	13.8	−15.2	1.6	−19.2	6.72

DISCUSSION OF DATA ON FERROMANGANESE PRACTICE.

GENERAL DESCRIPTION OF FERROMANGANESE PRODUCTION.

No specific information on the production of ferromanganese in the blast furnace is available in technical literature. Certain principles are generally understood to be true; these probably being: (*a*) The blast temperature should be high, (*b*) the slag should be basic, (*c*) the hearth temperature should be high.

HIGH BLAST TEMPERATURE.

Table 17 shows that temperatures as low as 735° F. have been used. The temperature for all 40 runs was only 1,135° F. In 1876, ferromanganese was made at the Diamond furnace, Cartersville, Ga.[a] A cold blast driven by a water wheel through a single 3-inch tuyère was employed. In making a ton of 55 per cent ferromanganese, 6,050 pounds of siliceous native ore, with 35 per cent Mn, was used, indicating a 58 per cent recovery. Comparing this with period 22, furnace J, Table 12, where a manganese recovery of only 55 per cent was made with 1,100° F. blast temperature and a 35 per cent ore, it will be seen that whatever economy results from the hot blast, it has not been found at all necessary to actual and profitable operation.

BASIC SLAG.

The average ferromanganese slag represented in Table 17 has the composition 41.7 per cent base, 14 per cent alumina, 28.1 per cent silica, and 10.6 per cent manganese. The average pig-iron slag[b] runs 48.7 per cent base, 13.1 alumina, and 35.3 silica. If a "more basic slag" means that the percentage of base is greater, then the ferro slag is 7 per cent less basic than the pig-iron slag. This method of comparison, however, is hardly proper. The function of the ferro furnace is to reduce the MnO from the slag and leave a practically irreducible calcium and magnesium aluminosilicate. This reduction is never complete. What actually exists may be considered to be reducible MnO as a solute dissolved in the quarternary mixture of irreducible oxides (CaO, MnO, Al_2O_3, SiO_2) as a solvent. It is the composition of the solvent that will affect the extent of the MnO reduction reaction. Compared on the basis of base + alumina + silica = 100 per cent, the two slags analyze as follows:

Analysis of pig-iron slag and of ferromanganese slag.

		Pig iron.	Ferromanganese.
Bases	per cent..	50.3	49.8
Alumina	do....	13.4	16.7
Silica	do....	35.3	33.5

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[a] Ward, W. P., Manufacture of ferromanganese in the blast furnace: Trans. Am. Inst. Min. Eng., vol. 5, February, 1876, p. 611.

[b] Feild, A. L., and Royster, P. H., Slag viscosity tables for blast-furnace work: Tech. Paper 187, Bureau of Mines, 1918, p. 17.

The percentage of bases is identical for the two slags. The only difference is that the ferromanganese slag carries a little more alumina and a little less SiO_2. This difference, moreover, is due to the character of the materials used and is not the result of any metallurgical design.

HIGH HEARTH TEMPERATURE.

At furnaces A, B, F, I, J, and K, temperatures were taken by means of several Leeds & Northrup optical pyrometers, with telescope sighted (1) through the tuyères both with and without a tuyère glass interposed; (2) on the surface of the slag at flush; and (3) on the surface of the metal at cast. As was to be expected, the temperatures observed opposite the tuyères varied over a rather wide range. To indicate the normal temperature variation in practice, the following comparison was made: Two hundred individual readings taken at seven furnaces, chosen to give a minimum, averaged 1,438° C. These measurements at each furnace were read by at least two observers, with either two or three separate pyrometer sets, through two or more tuyères, and at five well separated periods of time scattered through two to five days. The same sort of an average selected to give a maximum read 1,653° C. The difference between the maximum and minimum here can not well be attributed to any unusual furnace operation or to errors of observations or to instrumental errors. In the face of these figures, it must be accepted that as furnaces are run at present, a day-to-day variation of 225° C. in hearth temperature is not abnormal.

Laboratory tests strongly indicate that 1,265° C. is the lowest temperature at which MnO is reducible by either carbon or carbon monoxide. The lowest practical hearth temperature therefore may safely be set at 1,350° C. Superpose a variation of 225° C. on this minimum and the limits read 1,350° and 1,575° C. The maximum tuyère readings at the hottest furnace observed averaged 1,750° C. The minimum readings at the coldest furnace observed averaged 1,375° C. Where a temperature phenomenon such as this covers almost the whole range of practicable metallurgical temperatures, there is no room left on the temperature scale which the ferro furnace can monopolize as exclusively its own. Average temperatures taken on these seven furnaces were approximately as follows, the tuyère readings being corrected for the absorption by the tuyère sight glass when present and the temperature of the metal and slag corrected for emissivity as given by Burgess:[a]

	°C.
Temperature opposite tuyères	1,550
Temperature of metal at cast	1,386
Temperature of slag at flush	1,426

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[a] Burgess, G. K.: Temperature measurements in Bessemer and open-hearth practice: Bureau of Standards Technologic Paper 91, 1917, pp. 8-9.

RÉSUMÉ OF CONDITIONS IN FERROMANGANESE FURNACE.

It has been shown that in changing a furnace over from iron to ferromanganese it is neither necessary nor usual to raise the blast temperature, to increase the basicity of the slag, or to run with a hotter hearth.

True, there are certain resultant changes in the practice, the most important one being that the coke requirements are tripled. The daily tonnage is reduced to about one-third; the top temperature rises to about 1,000° to 1,500° F.; the carbon monoxide content of the top gas is increased about 50 per cent; and the blast pressure drops to 3 to 8 pounds per square inch. The operation of the furnace becomes easier; stock descends without hanging, slips become infrequent, fewer tuyères burn out, sulphur is taken care of completely as MnS in the slag, and further, there are no silicon requirements to be met. All furnace difficulties practically resolve themselves into one problem—to get into the metal the maximum percentage of the manganese charged.

THEORY AS TO FUEL REQUIREMENTS FOR PRODUCING FERROMANGANESE.

A number of heat balances with values in B. t. u. have been published on the iron furnace, but it seems hardly justifiable to construct a heat balance for the ferro furnace for comparison, for the following reasons: At room temperature, the heat of combustion of $C+O_2$ to CO_2 is either 4,250 or 4,430 B. t. u. per pound of C; at 1,500° C. it is either 4,550 or 4,950 B. t. u., depending on the authority selected. There is a wide lack of agreement among the few existing determinations of the heat of combustion, and greater lack of agreement among the experimental values for the specific heat of carbon, of oxygen, and of CO. At 600° C. the specific heat of carbon may be 0.31 or 0.44. At metallurgically interesting temperatures (between 1,400° and 1,800° C.), the discrepancy in figures for the total heat of carbon is about 30 per cent.

The specific heat of CO and of N_2 at 1,600° C. may be either 0.266 or 0.298. For the iron furnace the heat imparted to the gaseous products of reaction per pound of metal is either 972 or 1,280 B. t. u. The rise in temperature of gases above that of the burning coke is, therefore, either 696° or 1,070° F., 54 per cent discrepancy. The continued reprinting of some arbitrarily selected figure for the necessary thermal constants is both practically and scientifically objectionable and tends almost invariably to erroneous conclusions.

It is possible, however, to give an idea of the reason for the difference between the fuel requirements of the iron and those of the ferromanganese furnace and yet avoid excessive accumulation of meaningless B. t. u. figures. The comparison following is of interest.

Comparative operating data for pig iron furnace and for ferromanganese furnace.

	Pig iron.	Ferromanganese.
Carbon charged, pounds	1,727	5,524
Stone charged, pounds	984	2,349
Slag, pounds per ton	1,160	3,196
Gas analysis:		
CO_2 (by weight), per cent	21.43	10.44
CO, per cent	23.13	31.00
N, per cent	55.02	58.36
Blast temperature, °C	555	613
Top temperature, °C	204	363

The figures for the iron furnace are the averages for eight of the largest blast furnace plants in the country. The figures for the ferro furnace are the averages from Table 17. The calculations of the weight of the blast and of the gas and the determination of the amount of carbon absorbed by CO_2 and that burned at the tuyères for the ferro furnace are given in Table 19 following.

TABLE 19.—*Selected data on operation of ferromanganese blast furnaces.*

	Pounds.
Carbon charged per ton of alloy	5,523
Carbon combined with metal (6.5 per cent C)	145
Carbon blown out of stack as dust (9.6 per cent assumed)	530
Carbon available as fuel	4,848
Carbon from CO_2 in stone	263
Total carbon gasified	5,111
Carbon in top gas by weight, per cent	0.1615
Top gas weight (per ton of alloy: 5,111÷0.1615)	31,647
Weight of N_2 in top gas: (0.5836×31,647)	18,469
Weight of air blast: (18,469÷0.7672)	24,073
Weight of O_2 in blast: (24,073×0.2310)	5,561
Weight of carbon burned at tuyères: (5,561÷1.333)	4,172
Carbon absorbed in stack (4,848−4,172)	576

The two most important thermal reactions in the blast furnace are:

(1) $C+O=CO+4{,}450$ (or 4,950) B. t. u. per pound of C (at 1,600° C.).

(2) $CO_2+C=2CO+5{,}410$ (or 5,900) B. t. u. per pound of C (at 850° to 1,300° C.).

In reaction 1 both the carbon and the oxygen are supposed to be at the temperature of the solid stock in the combustion zone. As already shown this temperature is probably 1,550° C. for the ferro furnace and 1,590° C. for the iron furnace. Subtracting from the total heat of the reaction sufficient heat to bring the blast up to the temperature of the solid stock in the combustion zone gives a net heat to the gaseous product of reaction of 972 (or 1,280) B. t. u. per pound of metal made for the iron furnace and of 3,690 (or 4,800) B. t. u. for the ferro furnace.

In the ferro furnace, either in the hearth, at the tuyères, or just above them all the manganese in the metal was reduced from MnO. All of the CO_2 resulting from this reduction is itself reduced by solid carbon to CO; there is involved in this reaction 0.218 pound of carbon per pound of metal, resulting in a loss of 1,180 (or 1,290) B. t. u., leaving 2,500 to 3,500 B. t. u. to be used in heating the ascending gas above the temperature of the solid stock. The ferro-furnace gas weighs 12.6 pounds per pound of metal and the iron-furnace gas 4.2 pounds. Hence, in the ferro furnace the amount of heat available for raising the temperature of the gases is two to three times greater than in the iron furnace, and the weight of gas to be heated is three times as great. The rise in temperature, therefore, is either the same or less in the ferro furnace than in the iron furnace.

It is unwise to go here into any detailed consideration of why it is necessary to heat the ascending gas to this rather definite but somewhat unknown elevation above the temperature of the descending stock. It may be pointed out, however, (*a*) that some 250 tons of solid stock must be heated from 70° F. to above 2,800° F. in 24 hours; (*b*) that the greater part of this stock, being coke and mineral oxides, has a low thermal conductivity; (*c*) that the ascending gas stream does not have a chance to flow around every separate piece of coke, stone, and ore; and (*d*) that the transfer of heat from gas to solid is, under the most favorable conditions, a slow process. In order to heat the stock in the required time, therefore, the gases must at every point in the furnace be appreciably hotter than the stock; and this means not several degrees hotter but several hundred degrees hotter. Thus, the 4,165 pounds of carbon burned at the tuyères in the ferro furnace gives the gas stream no greater temperature elevation above the solid stock than does the 1,386 pounds burned in the iron furnace.

COMPOSITION OF FURNACE GAS.

At five furnaces samples of downcomer gas were taken and analyses made for the CO and CO_2 contents. The results obtained were somewhat variable but indicated a distinct tendency toward what might be called a normal composition.

As regards oxygen the evidence was that the true furnace gas contains no oxygen. Well taken samples showed 0.0 to 0.2 per cent. It was felt, however, that this oxygen could be safely attributed to leakage of air into the sample. Where the conditions of sampling permit the chance of contamination by the atmosphere, the oxygen content might be as high as 1.0 per cent.

The CO_2 content varied between the extreme limits of 11.6 and 3.5 per cent, with averages for the five furnaces of 6.7, 7.6, 7.1, 5.9, and 6.0 per cent. The CO content varied from 25.0 to 38.0 per cent, with averages for the five furnaces of 31.8, 34.2, 30.9, 30.0, and 31.0 per cent. The average analysis for the five furnaces was CO_2, 6.67 per cent; CO, 31.7 per cent.

Moissan[a] records an analysis of gas from a furnace making 60 per cent ferromanganese as CO_2, 5.5 per cent; CO, 30.0 per cent. This run, made in 1876, was one of the earliest attempts at the production of manganese in blast furnaces.

The analyses of the gas from nine of the largest iron blast furnace plants in the country show the following CO and CO_2 percentages:

Percentages of CO and of CO_2 in gas from nine iron blast furnace plants.

	CO_2	CO.
Furnace 1	12.6	26.0
Furnace 2	13.2	25.4
Furnace 3	13.0	24.3
Furnace 4	16.3	22.8
Furnace 5	14.3	24.3
Furnace 6	14.9	23.5
Furnace 7	15.6	23.2
Furnace 8	14.6	24.9
Furnace 9	13.3	25.4

The average complete analysis of the nine samples is:

Average analysis of gas from nine iron blast furnaces.

	Per cent by volume.	Per cent by weight.
CO_2	14.34	21.43
CO	24.32	23.13
CH_4	.36	.20
H_2	3.16	.22
N_2	57.82	55.02

Assuming from lack of better information that the H_2 and CH_4 content of the gases of the ferro furnace is the same as that of the iron furnace, and estimating the N_2 by difference, the analysis by volume and by weight of ferro gas is:

Assumed average analysis of gas from nine ferro furnaces.

	Per cent by volume	Per cent by weight.
CO_2	6.67	10.44
CO	31.17	31.00
CH_4	.36	.17
H_2	31.6	.21
N_2	58.64	58.18

"DIRECT" AND "INDIRECT" REDUCTION.

In Table 19, the weight of the furnace gas is calculated to be 31,647 pounds per ton of metal, which means, from the above analyses, 3,300 pounds of CO_2. The CO_2 from 2,341 pounds of the stone is 968 pounds, from the reduction of 190 pounds of Fe_2O_3, 156 pounds, and from the reduction of 3,770 pounds of MnO_2 to MnO by CO, 1,885 pounds; total, 3,009 pounds. This indicates a 9.56 per cent CO_2 content in the gas as against 10.44 per cent given on page 121. There is left 291 pounds of CO_2, which may be due to reduction of MnO by CO in the lower part of the furnace, to carbon deposition from CO in the upper part of the furnace, or—more likely—to experimental error either in the analyses, or in their application to the average practice given in Table 17. The evidence points closely to a 100 per cent reduction of MnO "directly" by carbon.

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[a] Moissan, H. H., Le manganese et ses composes, Encl. Chimique, 1886, p. 26.

It is well to point out that "direct" reduction by carbon probably means nothing of the kind. As in most important metallurgical problems, no experiment yet made settles this question. It can be assumed that the reaction:

$$MnO + CO = Mn + CO_2 \qquad (A)$$

is the reaction by which the final reduction of MnO takes place at a temperature somewhere between 1,265° and, say, 1,400° C. At this temperature the following reaction takes place, almost to completion, and with great rapidity:

$$CO_2 + C = 2CO \qquad (B)$$

Thus the result of the two reactions going on simultaneously, although not necessarily in the same part of the furnace, gives the same products as the single reaction:

$$MnO + C = Mn + CO \qquad (C)$$

As A and B together are not distinguishable from C by any calculation based upon gas analysis, one must be content to realize that either of these two reactions may be the one that takes place.

BURDENING AND DRIVING THE FURNACE.

It is all very well to apply to the furnace records accepted metallurgical theory, but practically the important thing is to have at hand something definite in the way of a guide to burdening and driving the furnace. The following discussion, therefore, is based exclusively on the figures in Table 17 and is free from any theoretical considerations whatever. It is an attempt to establish from the whole mass of data certain relationships between the 23 items of each run. There are only three factors that the furnace operators can change at will. They are (1) ratio of coke and ore; (2) ratio of stone and ore; and (3) quantity of wind blown per minute. It is assumed here that the blast temperature is kept as high as stove conditions permit. The number of rounds charged and the tons made per day are decided by the furnace itself. With the above three factors determined, there is nothing left for the operator to do but watch the furnace.

FUEL REQUIREMENTS.

Aside from the relatively constant amount of fuel needed to preheat, reduce, and melt the metal, the amount of carbon fuel required per ton of alloy should depend upon the temperature and humidity of the blast, the weight of the slag, and the temperature maintained in the combustion zone and hearth. That is, there should exist a relationship between five variables. In an attempt to determine this relationship from the data taken, the question of humidity must be dismissed from consideration, not necessarily because it is an unimportant factor but because no information is available concerning it. In the matter of hearth temperatures, no direct measurements are at hand; the pyrometer observations made by the bureau's investigators covered too short a period to prove of material assistance at this stage of the investigation. In a general sense the silicon in the alloy may be regarded as a thermometer, but for a given hearth temperature the amount of silica reduced to silicon undoubtedly depends on a number of other factors, including probably the composition of the slag and the speed of operation. The best indication of the temperature of the hearth is given by the amount of unreduced MnO in the slag.

The following procedure was adopted: The results of the 40 runs were arranged in order of decreasing slag weight and placed in four groups. The averages by groups appear in the first part of Table 20. The results were also arranged in the order of decreasing blast temperature, being again placed in four groups and the results averaged by groups. The averages are given in the second part of Table 20 following.

TABLE 20.—*Results of furnace runs arranged to show variation of carbon fuel with weight of slag and with blast temperature.*

VARIATION WITH WEIGHT OF SLAG.

Slag per ton.	Carbon per ton.	Blast temperature.	Carbon reduced to blast temperature of 1,130° F.
Pounds.	*Pounds.*	*° F.*	*Pounds.*
4,728	6,410	990	6,224
3,334	6,150	1,080	6,083
2,604	4,830	1,210	4,936
2,120	4,550	1,220	4,670

VARIATION WITH BLAST TEMPERATURE.

Blast temperature.	Carbon per ton of alloy.	Slag per ton of alloy.	Carbon reduced to 3,250 pounds of slag.
° F.	*Pounds.*	*Pounds.*	*Pounds.*
1,358	4,670	2,700	5,160
1,210	5,160	3,000	5,380
1,077	5,680	3,400	5,530
862	6,400	3,900	5,820

By the method of successive approximation it was found that 1 pound of extra slag requires 0.88 pound of carbon, and 1° F. extra blast temperature saves 1.33 pounds of carbon. Written in the form of an equation:

$$Ka=4170+0.88\ V-1.33\ T \qquad (1)$$

Where

Ka=pounds of carbon per ton of alloy used by the average furnace,
V =pounds of slag per ton of alloy,
T =blast temperature in degrees F.

This equation shows, for a given slag weight, for a given blast temperature, and for average humidity, the quantity of carbon required to heat the materials in the hearth to the average hearth temperature. As has been shown, this temperature is probably about 1,550° C., but it is better not to place too much confidence in this figure.

It is of course possible to charge either more or less carbon than the amount given by equation *1*. The average hearth temperature used by any given furnace operator means merely that temperature found by experience, or assumed from lack of experience, to be the most desirable to maintain. If the operator decides to use a higher temperature in practice, the furnace can be given "excess carbon," a phrase taken to mean the carbon actually charged minus the amount indicated by equation *1*. When excess carbon is used, the percentage of manganese in the slag will be lower, provided all the other conditions remain constant. On the other hand, the carbon charged may be considerably lower than that given by equation *1*, with a resultant hearth temperature lower than the average.

PERCENTAGE OF MANGANESE IN SLAG.

Determination of the percentage of manganese that goes into the slag is approached in the same manner as the problem of fuel requirements, with the exception that there must not be any "left over" variable such as excess carbon over requirements. The operator can at his own free will charge so little fuel that the hearth freezes, or he can charge so much fuel that he bankrupts the owners. The percentage of manganese in the slag, however, is the result of a number of metallurgical phenomena and is definitely determined by the actual furnace facts.

The data in Table 17 show that the percentage of manganese in the slag depended upon the following factors:

(1) The ratio of bases to silica in the slag;

(2) The hearth temperature as determined by the excess or deficiency of fuel above or below that indicated in equation *1*;

(3) The rate of driving.

RATE OF DRIVING.

The expression found most nearly to represent the speed of operation was the pounds of gross slag produced per minute per square foot of hearth area. Gross slag here means the weight of slag-forming materials, manganese being a slag-forming material; that is, the weight of slag that would be flushed from the furnace if none of the manganese was recovered in the metal. There is an implication here—probably warranted—that the MnO_2 charged is reduced to MnO in the upper stack, but that the MnO entering the bosh is fused with the silica, alumina, and bases, and that it is reduced to metallic manganese only after it has descended molten into the hearth. The calculation of the gross slag is simple. To take the figures from the average of the 40 runs represented in Table 17, as an example, the slag weight is 3,196 pounds. The metallic manganese in the metal is 1,680 pounds. This metallic manganese was reduced from 2,170 pounds of MnO. The gross slag was therefore 3,196 plus 2,170, or 5,366 pounds per ton of metal.

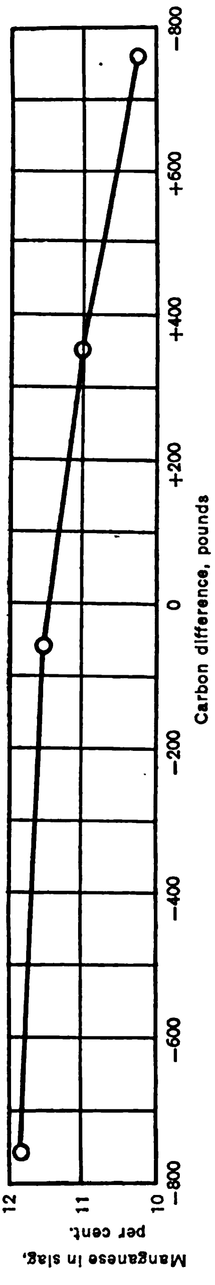

FIGURE 10.—Curve showing relation of percentage of manganese in slag to carbon difference (carbon charged minus carbon required for average hearth-temperature conditions; see Table 17).

It is well to notice that the more usual expression for speed of operation in blast-furnace parlance is the pounds of carbon burned per minute per square foot of hearth area. Without commenting upon this point of view as it applies to the iron furnace, its applicability to the ferromanganese furnace is extremely doubtful. The only factor limiting the speed of operation, assuming that the speed is not so great that the stock is blown out the top of the furnace, is the ability of the hearth to reduce manganese from the slag. As long as the hearth temperature is above 1,265° C., all the manganese in the slag should be reduced, provided the slag remains for a sufficient length of time in the hearth under the reducing influence of a CO atmosphere and in contact with solid carbon. Therefore, no principle of common sense is violated in taking as the criterion of speed the figure for the pounds of slag passing into the hearth from the bosh per minute per square foot of hearth area.

EQUATION FOR PERCENTAGE OF MANGANESE IN SLAG.

The results of the 40 runs represented in Table 18 were arranged in order of decreasing "excess carbon" as determined by equation 1. The results were then divided into four groups and the averages by groups recorded as in Table 21. The results are also plotted in figure 10. The results of the same runs were rearranged in the order of decreasing basicity (ratio of bases to silica), grouped, and averaged, and the averages recorded as in Table 22. The results are also plotted in figure 11. In the same way Table 23 gives the averages for groups taken in order of decreasing rates of driving. The results are also plotted in figure 12. The three tables follow.

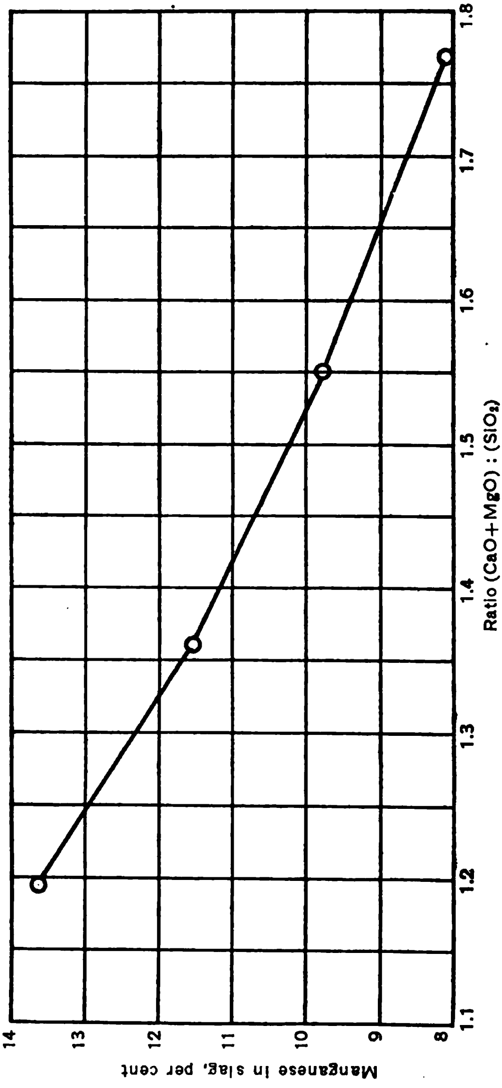

FIGURE 11.—Curve showing relation of percentage of manganese in slag to ratio (CaO+MgO): (SiO$_2$).

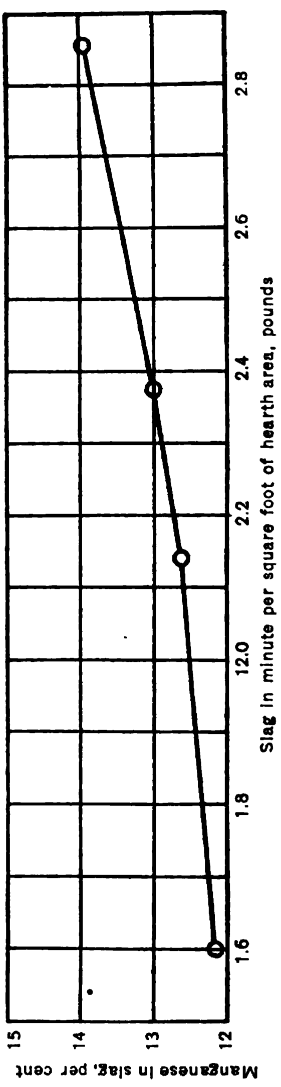

FIGURE 12.—Curve showing relation of percentage of manganese in slag to rate of driving (weight of slag made per minute per square foot of hearth area).

TABLE 21.—*Results of furnace runs arranged to show variation of percentage of manganese in slag with excess or deficiency of carbon fuel as required for average hearth-temperature conditions.*

[Results are plotted in figure 10.]

Group.	Runs included.	Furnaces included.	Carbon excess or deficiency (pounds.)	Basicity ratio.	Rate of driving.	Percentage of manganese in slag.	
						Actual.	Reduced to standard basicity[a] and rate of driving.[b]
1	4, 26, 30, 31, 32, 33, 37, 38, 40.	A, C, K, L, M, P..	+754	1.69	1.84	8.2	10.3
2	3, 7, 8, 12, 13, 14, 17, 25, 29, 39.	A, B, F, K, M....	+354	1.48	1.92	11.0	11.0
3	2, 5, 6, 10, 11, 19, 23, 24, 25...	A, B, G, J........	− 56	1.45	2.18	12.1	11.4
4	1, 15, 18, 20, 22, 27, 34.......	A, F, G, I, J, K, L.	−765	1.37	2.36	13.3	11.8

[a] Standard basicity (ratio of bases to silica), 1.50.

[b] Rate of driving (pounds of gross slag produced per minute per square foot of hearth area), 2.1.

TABLE 22.—*Results of furnace runs arranged to show variation of percentage of manganese in slag with basicity of slag (ratio of $CaO + MgO$ to SiO_2).*

[Results are plotted in figure 11.]

Group.	Runs included.	Furnaces included.	Basicity ratio.	Carbon excess or deficiency (pounds).	Rate of driving.	Percentage of manganese in slag.	
						Actual.	Reduced to standard hearth temperature and rate of driving.
9	6, 8, 23, 24, 25, 26, 28, 31, 33..	A, J, K, L........	1.77	+411	1.79	7.4	8.2
10	7, 11, 17, 30, 32, 34, 35, 39, 40.	A, B, F, L, M.....	1.55	+187	2.06	9.2	9.9
11	1, 4, 10, 12, 13, 20, 27, 38.....	A, B, C, I, K......	1.36	− 10	2.32	11.8	11.5
12	2, 5, 8, 14, 15, 19, 22.........	A, B, F, G, J......	1.19	−110	2.42	14.2	13.6

TABLE 23.—*Results of furnace runs arranged to show variation of percentage of manganese in slag with rate of driving (pounds of gross slag per minute per square foot of hearth area).*

[Results are plotted in figure 12.]

Group.	Runs included.	Furnaces included.	Basicity ratio.	Rate of driving.	Carbon excess or deficiency (pounds).	Percentage of manganese in slag.	
						Actual.	Reduced to standard basicity and temperature.
5	1, 4, 6, 7, 14, 15, 18, 22, 39....	A, B, F, J, M.....	1.38	2.86	− 60	13.0	13.9
6	2, 5, 8, 19, 20, 21, 25.........	A, G, I, J.........	1.70	2.38	−312	12.0	12.9
7	3, 10, 11, 12, 19, 23, 27, 40....	A, B, G, J, K, M..	1.51	2.15	− 62	12.6	12.5
8	13, 15, 17, 24, 26, 28, 38......	B, C, F, J, K......	1.64	1.60	+240	10.3	12.2

In deriving an equation for the percentage of manganese in the slag four quantities are involved. If only figures could be selected wherein two of the quantities varied while the other two quantities remained constant, the derivation of the equation would be direct and simple. Unfortunately this is not possible. Thus, in Table 22 there is a distinctly apparent relation between "carbon excess or deficiency," basicity, and rate. As the carbon decreases the basicity decreases and the rate of driving increases. This relationship is due to the human management of the furnace. The increase of basicity is governed by the stone charged; the carbon excess or deficiency by the coke charged; and the rate of driving by the speed of the engines. For any run the stone can be decreased, the coke increased, and the engines slowed down by practically any arbitrary amount, and the furnace will still continue to make ferromanganese, though it may not make a profit. The fact, therefore, that this apparent relationship exists between excess carbon, basicity, and speed, shows only that the furnace operator followed, perhaps unknowingly, a definite rule in burdening and driving the furnaces.

Although in Tables 21, 22, and 23 all the four quantities concerned vary, it is found by successive approximation that—

(*a*) 100 pounds of excess carbon lowers the manganese in the slag 0.11 per cent;

(*b*) An increase in the rate of driving of 1 pound of slag per minute per square foot of hearth area increases the manganese in the slag 1.34 per cent;

(*c*) An increase in the ratio of base to silica of 1 unit decreases the manganese in the slag 9.16 per cent.

In Table 21 values for the actual percentages of manganese in the slag are given in the seventh column. In the last column the values have been corrected for each of the four groups for the amount by which the basicity ratio differs from 1.50, using the correction indicated in (*c*); the values are further corrected for the amount by which the rate of driving differs from 2.1, using the correction indicated in (*b*). In the same way for Tables 22 and 23 the percentages of manganese in the slag are given in the last column corrected for variations of the two quantities, in the fifth and sixth columns, in order to show the relationship that would be found between the variable shown in the fourth column and the manganese in the slag, were the other two variables held constant. These relationships are plotted in figures 9, 10, 11, and 12. Combining these three relations in a single equation gives the result—

$$M = 21.4 - 9.16B + 1.34R - 0.0011(K - K_a) \quad (2)$$

where

M = percentage of manganese in the slag;
B = ration of base to silica;
R = rate of driving (pounds of gross slag per minute per square foot of hearth area);
K_a = carbon required by equation *1*;
K = actual carbon charged.

Combining equation *2* with equation *1* gives—

$$(M=21.4+1.34R-9.16B-0.0011\mathrm{K}-\mathrm{K}_a+0.00097V-0.00145T \qquad (3)$$

Equation *3* is the general solution for *M*, the percentage of manganese in slag, *M* being a function of five variables. The assumption has been made in deriving this equation that the function was linear with respect to all five variables. This certainly is not true. The effect of speed on the reduction of MnO will be, itself, a function of the temperature. Moreover, a large number of variables undoubtedly affecting *M* have been omitted, for example, the physical character of the stock, the temperature of the top gas, the amount of carbon absorbed by CO_2, and the humidity of the blast. It can be hoped only that equation *3* includes the more important variables.

It will be noticed that in joining the points in figures 10, 11, and 12, a curved rather than a straight line is drawn in each figure. This was done with the hope of giving a hint as to the nature of the curve. In each figure a straight line can be drawn which will not miss the experimental point so much as 0.5 per cent. In view of the limitations of the data 0.5 per cent is much inside the limit of experimental error.

BURDENING THE FURNACE.

Observance of the relations in equations *1*, *2*, and *3* may be of material assistance in the burdening and driving of the furnace. The first step in solving this problem is the calculation by equation *1* of the ratio of coke and stone to ore with an assumed basicity ratio. Then by looking at figures 10, 11, and 12, the furnace operator can know at a glance the change in the manganese content of his slag to be expected from adding to, or trimming down, his fuel, or from changing his basicity ratio. No general principles can be given for the solution of this problem as the most important considerations governing the burdening problem are the cost of the materials used, the daily pay roll, and the furnace overhead. Every change in the burdening and driving factors cuts both ways. Slow driving reduces slag loss and runs up labor and overhead charges. Increasing the proportion of lime in the slag within reasonable limits results in lowering the slag loss, but increases coke consumption, and cuts down tonnage, resulting in lower ore cost, higher fuel cost, and increased labor and overhead charges. Using excess coke results in a slight decrease in the percentage of manganese in the slag, but in general increases the slag weight more rapidly, so that the slag loss, being the product of the slag weight and the percentage of manganese in the slag, is increased. About the only general criticism that can be made of the burdening and driving data is that there is on the whole a tendency to use too much fuel and to drive too slowly. The saving of ore resulting from slow driving, while appreciable, causes a decided decrease in profits of the furnace and, although in line with conservation of materials, will not appeal to the operator who is to pay the labor and overhead charges.

It may be stated that the most effective methods of reducing the slag loss of manganese are: (1) The use of low-ash coke and of low-silica stone and (2) carrying as basic a slag as is practicable. In the matter of selecting a low-ash coke it was not possible, perhaps, in the past year for a number of operators to do more than take whatever they could get.

It was universally realized at the furnace that the presence of a large percentage of silica in the coke was playing havoc with the practice; but it is believed that, had the office fully appreciated the loss in profits occasioned by excess silica a considerable improvement in coke could have been effected. Actual purchases of stone, it is certain, have not been made with the care the problem merited. Calcites and dolomites have been used with silica content as high as 7 per cent. Naturally the office did not purchase this stone on the market without a considerable preferential price, but, as is shown later, stone with this silica content should not be used even if obtained free of charge f. o. b. the furnace.

STACK LOSS.

The sum of the manganese in the slag and that in the metal does not equal that charged into the furnace. The percentage of manganese in the charge not accounted for by the manganese carried from the hearth is for convenience called "stack loss." The value of this quantity in Table 17 varies from 0.8 to 32.1 per cent. Its average is 12.8 per cent. As the average loss of manganese in the slag is only 14.7 per cent, the stack loss is as important as the slag loss. It is a quantity, however, that has not thus far proved itself susceptible to analysis or definite explanation. It may be due to: (1) Volatilization of metallic manganese from the hearth, (2) volatilization of manganese oxide in the hearth or stack, (3) manganese oxide carried off mechanically as ore fines by the blast, and (4) a fictitious result due to error in chemical analyses and weights.

To a certain extent in any individual run the apparent stack loss is surely a combined result of these four causes. But for the 40 runs taken as a whole it is difficult to understand how errors in weights and analyses would give a *loss* in 39 cases, instead of giving for one-half of the runs a "*gain* of manganese in the stack." Unless it can be shown that the analyses for manganese in the ore are systematically high or those for manganese in the slag and metal are systematically low, it must be concluded that the errors are sufficiently eliminated in the average of a number of runs.

There is a possibility that the loss as ore fines carried out mechanically by the blast may account for the whole loss.

The operating data from nine iron furnace plants indicate that 6.5 per cent of the iron charged is not accounted for by the iron in the slag and metal. This figure does not inspire much confidence and is given here only for the sake of comparison. As to the loss by volatilization, it is probably not important practically to distinguish between volatilization of metallic manganese and volatilization of manganese oxide. The answer to this problem will appear doubtless, when the relation is found between stack loss and some other operating quantity. If the stack loss is a volatilization phenomenon, it should increase with a rise in hearth temperature. It should also decrease with any increase in the speed of operation, as the longer a given pound of slag or metal remains in the hearth the greater the volatilization. The 40 runs in Table 17 were arranged in the order of decreasing stack loss and divided into four groups. The average for each of the four groups appears in Table 24 following:

TABLE 24.—*Results of furnace runs arranged to show manganese lost in stack as related to other factors.*

Group	1.	2.	3.	4.
Mn lost from stack, per cent	24.4	15.7	9.1	4.0
Top temperature, °F	994	851	995	904
Mn in slag, per cent	8.6	13.5	11.1	10.6
Si in metal, per cent	1.43	.95	1.15	1.26
Ratio of base to silica	1.61	1.47	1.56	1.49
Rate of driving [a]	2.17	2.32	1.95	2.23
Carbon excess or deficiency, pounds	+253	−86	+82	−121

[a] Pounds of gross per minute per square foot of hearth area.

The results presented in the table show a manganese loss for each group of runs, but no systematic variation of stack loss with any other probably related quantity. High hearth temperatures as indicated either by high-silicon metal, excess carbon, basic slag, or low percentage of manganese in the slag appear not to affect the amount of stack loss. There is only one practical conclusion to be formed—the furnace man must admit that the stack loss is an important physical constant and should operate his furnace on that assumption, or he must conclude that the data here presented are insufficient to permit a solution of the problem and obtain for himself more complete and exact furnace records.

SOME TYPICAL CALCULATIONS.

To illustrate the use of calculations suggested in this report, to indicate how many factors are involved in the simplest detail of furnace practice, and to serve as an introduction to the operating problem, a brief outline of the factors involved in the selection of coke and stone follows.

COST FIGURES FOR AVERAGE PRACTICE.

For the sake of comparison a computation of the approximate furnace costs for the average run represented in Table 17, apart from investment charges, may be of interest.

The average charge per ton of metal is 2.67 tons of ore, 3.16 tons of coke, and 1.03 tons of stone. At the arbitrary prices of $40, $10, and $3 per ton, f. o. b. furnace, the materials cost $149.49 per ton of metal made. The following daily costs are assumed: Six hundred dollars for labor, and $175 for the aggregate of superintendence, chemical laboratory, and reserve for relining, repairs, liability insurance, clerical work, demurrage, supplies, etc. Thus there is a total fixed charge of $775 against a daily production of 51.7 tons of ferromanganese, or a charge of $14.75 per ton. Hence the total cost per ton is $164.24. The selling price on 74.9 per cent ferromanganese is taken to be $267.59 ($250 per ton for 70 per cent alloy plus or minus $3.50 per unit of manganese), so that the profit per ton is $103.35, or $5,343 per day.

COKE ASH.

The following analyses of coke are taken from furnace records and although the coke represented is not typical of that used in average practice, it is not abnormal, as millions of tons of such coke have been used annually at certain iron furnaces for years.

Analyses of coke used in certain iron furnaces.

Sample No.	SiO_2.	Al_2O_3.	Fixed carbon.	Volatile matter.	Sulphur.	Moisture.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
1	2.78	2.47	90.95	0.79	0.41	1.35
2	1.80	2.70	91.12	.47	.44	1.52
3	3.06	2.43	91.06	1.18	.80	1.40

Let it be assumed that coke 1 is available for a ferro campaign and that stone of grade "A," as described in the section following ("Selection of Stone"), is also available. Let the ore have the average analysis given in Table 17. Then it is possible to construct a charge sheet for these materials to give approximately the maximum daily profits, subject to the limitation that none of the operating quantities shall lie appreciably beyond what has been tried out as shown by Table 17. The charge will be then as follows:

Analysis of charge.

Material.	Weight, pounds.	SiO_2.		Al_2O_3.		Bases.		Mn.		Fe.		C.	
		Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.
Ore	5,310	8.60	457	2.50	133			40.33	2,240	5.93	825		
Coke	4,000	2.74	109	2.47	98							91.95	3,620
Stone	2,102	.42	10	.13	3	53.0	1,115						
Total			576		234		1,115		2,240		325		3,620

The manganese balance will be as follows:

Manganese balance.

	Pounds.
Charged	2,240
Stack loss (12.66 per cent)	[a] 284
Slag loss (8.43 per cent)	[b] 189
To metal (79 per cent)	1,767
Accounted for	2,240

The weight of the slag will be 2,167 pounds, the analysis of the slag being as follows:

Analysis of slag.

Constituent.	Per cent.	Pounds.
SiO_2	26.05	[a] 561
Al_2O_3	10.85	234
Bases	51.75	1,115
MnO	11.30	241
S		17
Total		2,167

[a] 151 pounds to silicon in metal.

The carbon as required by equation *1* would be:

$$K=4,170+0.88\times 2,167-1.35\times 1,250=4,405$$

The "deficiency" of carbon charged below that required by equation *1* is therefore 785 (4,405 − 3,620) pounds, only 30 pounds less than was used in the runs in group 1, Table 21. According to equation *2*, this lack of fuel causes the manganese in the slag to increase only 0.87 per cent. Referring to figure 9 it will be seen that the actual curved line falls somewhat below this assumed straight line.

The ratio of bases to silica is 51.75 ÷ 26.05, or 1.99. This basicity is 1.99 − 1.48, or 0.51 higher than that of the average run, and lowers the percentage of manganese in the slag by 4.67 per cent, according to equation *1*. Figure 12 shows that the curved line lies above the straight line. To be conservative, a decrease of only 3.75 per cent is taken, the smallest value figure 12 will justify.

The gross slag is 2,167 + 1.29 × 1,777, or 4,485 pounds. For comparison, the hearth diameter of the furnace is taken to be 10.78 feet, which gives a hearth area of 91.3 square feet, the average hearth area in Table 16 if each furnace be weighted in the average according to the number of runs represented in the table. A driving rate of 2.61 pounds of gross slag per minute per square foot of hearth area is taken. This gives the output as 77.5 tons per day. The rate 2.61 is 0.54 greater than the average rate in Table 17, and, according to equation *2*, causes a rise in the percentage of manganese in the slag of only 0.71 per cent. Within this part of the curve in figure 11 it is difficult to take a greater rise than 0.75 per cent; which figure will be used.

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[a] Average.

[b] As shown below.

From the above, then, the percentage of manganese in the slag is—

	Per cent.
Average in Table 17	10.69
Increase due to low fuel	.87
Increase due to fast driving	.75
Decrease due to high basicity	3.75
Manganese in the slag	8.57

Slag loss (slag weight times percentage of manganese in slag: 2,167×8.75)	189 pounds.
Slag loss in percentage of manganese charged	8.43 per cent.

Summarizing:

Charge:	
Ore, 5,310 pounds, 2.48 tons, at $40	$99.38
Coke, 4,000 pounds, 2 tons, at $10	20.00
Stone, 2,102 pounds, 0.94 ton, at $3	2.82
Cost of materials	122.20
Daily labor and fixed charges, $775.	
Charge against 1 ton of metal, at 77.5 tons per day	10.00
Total furnace cost of 1 ton of metal	132.20
Selling price, $250 for 70 per cent ferro, with $3.50 per unit bonus and penalty.	
Selling price of 1 ton of ferro containing 79 per cent Mn	281.50
Furnace profit per ton	149.30
Furnace profit per day, at 77.5 tons per day	11,570.76
Furnace profit per day of furnaces represented in Table 2	5,323.00
Gain in profits due to improved materials and practice	6,247.76

These figures are disturbing, yet it is difficult to see just where they are wrong. The matter is important. If the 11 furnaces represented in Table 17 had operated on this average practice the possible saving would have been $2,061,760.80 per year per furnace, or, for the 11 furnaces, $22,679,368.80. The practical problem would have been to obtain the high-grade coke and stone involved.

An interesting point in the situation is that the iron furnace using the coke best adapted for the production of ferromanganese, namely, that designated as sample 2 in the table on page 112, used stone with 3.36 per cent SiO_2 (to increase the slag volume), and yet made metal of the following composition: Si, 1.52 per cent; S, 0.049 per cent; P, 0.173 per cent.

With the phosphorus content indicated, the metal is neither foundry nor Bessemer grade. As basic pig, 1.52 per cent silicon is objectionably high, unless a better sulphur content than 0.049 per cent can be attained. The answer is that the coke used was lower in SiO_2 than was really desirable. A good coke consumption and a fair tonnage can be shown with low-silica coke and stone, but in basic practice, where the logical function of the blast furnace is to keep down the sulphur and the silicon, the slag volume should not be so low that it can not take care of the sulphur.

Undoubtedly the operator of the basic furnace above mentioned realized this. Enough commercial pressure would have made him release this coke. If the ferro furnace had paid $20 per ton for low-silica coke, the daily profit would still have been $10,000 per furnace, an improvement of $4,523 per day, or roughly 100 per cent. If the desired quality of coke could have been procured for $20 per ton, and if the other assumptions are correct, the increased profit for any one furnace would have been $1,800,000 per annum.

SELECTION OF STONE.

As a typical problem relating to the selection of stone for use in furnaces producing ferromanganese, two grades of stone may be assumed to be under consideration. One may be designated grade A, containing 53 per cent basic constituents and 0.42 per cent silica; the other may be designated grade B, containing 49.5 per cent basic constituents and 7 per cent silica. Stone of grade A has been used in great quantities at one iron furnace plant, and hence is not of purely hypothetical composition. Stone of grade B has been used in producing ferromanganese.

If all the values, except for the grades of stone, be taken as for the average run represented in Table 17, a comparison can be made as follows:

Comparative results obtained with two grades of stone.

Constituent of charge.	Stone A.		Stone B.	
	Quantity per ton of alloy.	Cost.	Quantity per ton of alloy.	Cost.
	Tons.		*Tons.*	
Ore	2.64	$105.56	2.72	$108.80
Coke	3.12	31.15	3.54	35.45
Stone	1.05	3.15	1.70	5.09
Total	....	139.86	....	149.34
Manganese in metal, per cent	75.6		72.15	
Slag per ton of alloy, pounds	2,890		3,839	

The foregoing data are based on the assumption that the slag has a constant basicity ratio of 1.48 and that an increase of 1 pound in the rate of slag formation requires an increase in carbon of 0.88 pound.

The gross slag produced per ton of metal when stone A is used is 5,150 pounds, and when stone B is used 5,970 pounds. If the rate of driving be assumed to be constant, the tonnages will be 51.7 tons with stone A and 44.65 tons with stone B. The daily fixed charges will be roughly the same for both. There will, therefore, be a charge of $14.90 against a ton of metal if stone A be used and a charge of $17.35 if stone B be used.

The total costs are $154.61 and $166.66, respectively, per ton of alloy. The selling prices are $269.60 and $257.53; the profit per ton, therefore, when stone A is used is $114.99, and when stone B is used, $90.87. The profits per day are $5,950 and $4,060. Thus, the daily loss in furnace profits when stone B is used is $1,890. This loss must be charged to the 75 tons of B stone used per day. Thus, even if this grade of stone were obtained free of charge, the furnace loss from its use would be $22.20 per ton. Therefore, if stone A could be purchased at some premium less than $22.20 per ton, a furnace having no choice other than buying A or B would profit by the purchase of A. Such computations as these could be widely extended showing the actual premiums and penalties for coke and ore as based on the cost to the furnace, but the above example will suffice here to substantiate the following conclusions:

(1) The blast-furnace superintendent should know at all times the cost and the analyses of all materials purchased, and should also be familiar with all purchasable materials even including $40 coke and $20 stone.

(2) The decision as to the purchase of raw material should be left to the furnace superintendent, who should be in complete charge of the operation of the furnace.

(3) Far greater saving in furnace profits can be attained from judicious selection of raw materials than from any changes in operating conditions.

When the market price of the alloy falls, as it doubtless will, and the costs of stone, coke, ore, and labor, readjust themselves, it is only by a continued revision of some such calculations as the above, covering the coke and stone, that the furnace operator can hope to make the best showing.

SUMMARY OF OBSERVATIONS ON FERROMANGANESE DATA.

Operating data were collected from the furnace records of 11 blast furnaces making ferromanganese. These data are presented in Table 17, together with a maganese balance showing the distribution of the manganese charged between metal, slag, and top gas. The table includes 40 "experimental periods" of 10 days each and covers the extreme range of operating conditions for each furnace.

Several thousand pyrometer measurements were made of the temperature in the furnaces opposite the tuyères; of the slag at flush and cast; and of the metal at cast. It is shown that, in general, the ferromanganese furnace operates with a colder hearth and produces colder metal and slag than does the iron furnace.

The results of gas analyses made by the bureau show that practically all of the MnO is reduced "directly" by carbon. This reduction probably takes place in the hearth.

The metallurgical questions involved in the smelting of manganese are briefly discussed. It is shown that although more than three times as much carbon is burned at the tuyères in the ferro furnace as in the iron furnace, the gaseous products from the combustion zone are probably at a lower temperature.

The two losses of manganese in the furnace—the slag loss and the "volatilization" loss—are discussed. It is shown that the percentage of manganese in the slag is decreased by raising the blast temperature, by increasing the basicity of the slag, and by charging more coke; and that it is increased by fast driving, and by carrying a greater slag "volume." A general equation is given for the percentage of manganese in the slag as a function of these five quantities, by means of which the slag loss can be calculated in advance from the charge-sheet data.

The "volatilization" loss is shown not to be affected by the basicity of the slag, the silicon in the metal, the carbon charged as fuel, the rate of driving, or the temperature of the top gas. The results, however, are not convincing when examined carefully, and additional and more exact figures are needed. It is shown that by making five changes in practice the 11 furnaces examined could have raised their recovery to 79 per cent, using the same ore, could have increased their tonnage 59 per cent, and could have increased their annual net profits by more than twenty million dollars. The changes are as follows:

(*a*) Using a better grade of coke existing in sufficient quantities and probably purchasable.

(*b*) Using a better grade of stone.

(*c*) Using less fuel.

(*d*) Running with a more basic slag.

(*e*) Driving faster.

Certain furnaces are shown to have lost $10 to $20 a ton by using high-silica stone.

It is concluded that there is greater room for improvement in the selection of coke and stone than in furnace practice.

DISCUSSION OF DATA ON SPIEGELEISEN PRACTICE.

GENERAL DESCRIPTION OF SPIEGELEISEN PRODUCTION.

The simultaneous smelting of iron and of manganese ores—that is, the production of spiegeleisen—is metallurgically an uneconomical process. The reason can readily be appreciated. The iron furnace and the manganese furnace present problems that, though somewhat similar, are at many points fundamentally different. The most economical operation for making iron is not the most economical for making manganese. It is desirable, therefore, that the two processes be kept separate. The only logical reason, of course, for making spiegeleisen at all is to utilize manganiferous iron ores. Spiegeleisen has not, however, always been made in this way. The manganese content of the metal made is frequently reduced by the addition of iron ore or of iron scrap to the charge. During the year 1918, pressure from several sources was brought to bear upon the manganese producers to lower the manganese content of their alloys. The prevailing standard for ferromanganese was lowered from 80 to 72 per cent, and for spiegeleisen from 20 to 16 per cent. To the extent that this was done with the use of lower grade ores, it served a real purpose by creating a market for otherwise unsalable ores. It should be remembered, however, that such a process was wasteful of coke and manganese, and caused decreased furnace tonnage. The rather analagous method of "sweetening" a spiegel ore mixture with high-grade manganese ore is equally objectionable from the standpoint of a metallurgist.

The average spiegel analysis given in Table 18 (p. 116) shows that in order to produce 1 ton of manganese, 5.78 tons of spiegel was required. This alloy contained about 4.4 tons of metallic iron, the equivalent of 4.64 tons of pig iron. The carbon fuel required to make 5.78 tons of spiegel is 19,900 pounds (5.78×3444). Crediting this metal with the amount of carbon required for making 4.64 tons of pig iron, namely, 1,727 pounds of carbon multiplied by 4.64 tons of pig iron, or 8,000 pounds total, it is seen that the ton of metallic manganese in the form of spiegel requires 11,900 pounds of carbon.

It is clear that in 1.33 tons of 74.9 per cent alloy there is 1 ton of metallic manganese and 0.23 ton of metallic iron, the equivalent of 0.25 ton of pig iron. The fuel consumption was 7,100 pounds of carbon (1.33×5323) according to Table 17. Giving credit for 420 pounds (1727×0.25) for the iron made, 1 ton of manganese in the form of ferro requires 6,680 pounds of carbon. Thus, it appears that a ton of manganese in the form of spiegel requires 78 per cent more carbon fuel than a ton of manganese in the form of ferro.

This result is by no means wholly, or even largely, to the mere combination of the two smelting processes. Most of the excess fuel requirement is due to the character of the ferro ores and of the spiegel ores used. The manganese and silica contents of the two ores are:

	Spiegel.	Ferro.
Manganese, per cent	12.75	40.33
Silica, per cent	14.84	8.60
Silica per pound of manganese, pounds	1.16	.214

The average spiegel ores were therefore 5½ times as siliceous as the ferro ores.

SLAG COMPOSITION.

Table 18 shows that the average spiegel slag is less basic than either the iron slag or the ferro slag. The following figures will permit a comparison of these three average slags calculated on the basis of base + alumina + silica = 100 per cent.

Comparison of three average slags.

	Base.	Alumina.	Silica.	Ratio (Base to SiO_2).
	Per cent.	*Per cent.*	*Per cent.*	
Pig iron	50.3	13.4	35.3	1.42
Spiegel	47.9	14.1	38.0	1.26
Ferro	49.8	16.7	33.5	1.49

Table 18 demonstrates that basicity ratios higher than 1.26 have been successfully used in spiegel practice. In fact, the eight runs showing the highest basicity ratio, namely, Nos. 46, 57, 58, 61, 64, 66, and 67, including furnaces B, D, H, J, M, P, and G, show an average ratio of 1.42—exactly that of the slag in the average iron practice quoted above. The average recovery of manganese by the metal for these eight runs was 70 per cent.

It is of interest to compare these figures with those for the eight runs (42, 43, 50, 51, 52, 53, 54, and 60) which had the lowest basicity ratios, the average being 1.08. The average recovery for these runs was 47.3.

HEARTH TEMPERATURE.

At furnaces C, D, E, H, M, and Q, optical pyrometer observations were made of slag, metal, and tuyère temperatures. The same day-to-day variations in temperature were noted. The averages of all temperatures from the six furnaces are given below and the figures from the ferro-furnace observations are repeated for comparison:

Average temperatures in speigel and in ferro furnaces.

	Spiegel.	Ferro.
Temperature opposite tuyères, ° C	1597	1550
Temperature of metal at cast, ° C	1392	1386
Temperature of slag, ° C	1427	1426

Seemingly, both the metal and the slag flow from a spiegel furnace at the same temperature as from the ferro furnace.

COMPOSITION OF FURNACE GAS.

Analyses of spiegel furnace gas were made by the bureau's field party at only four furnaces. The composition of the gas as indicated by the average of all the observations was as follows:

Average composition of speigel furnace gas.

		By volume.	By weight.
CO_2	per cent..	7.08	11.05
CO	do....	30.68	30.40
N_2	do....	58.72	58.08

These figures are not at all satisfactory. The CO content is nearly as high as in ferromanganese-furnace gas. It is difficult to charge this result to ordinary analytical errors. Leakage of air into the sample would lower the CO content; also, the estimation of CO by cuprous chloride absorption when improperly carried out gives a result too low rather than too high. The average analyses for the four furnaces gave 4.62, 5.62, 8.95, and 9.24 per cent for CO_2 content and 24.42, 29.25, 34.20, and 34.85 per cent for the CO content. The extreme limits for CO_2 were 3.9 and 11.1 per cent; and for CO, 23.3 and 35.1 per cent. This range of variation is the same as that encountered in ferromanganese-furnace gas. This fact points strongly to the conclusion that in both of these alloy furnaces, the chemical reactions in the lower part of the furnace are continuously changing. It might almost be said to prove that for a period of time the bosh and hearth reactions are nearly those of the iron furnace, and that following this period as a result of the settling down of the stock, a large amount of manganese oxide is thrown into the hearth, resulting in a rapid reduction of MnO, with a consequent increase in the ratio of CO to CO_2. Therefore, unless more or less continuous samples of furnace gas are taken over a rather long period, the actual analysis might easily misrepresent the true average composition of the furnace gas.

CARBON BURNED AT THE TUYÈRES.

From what has been said in regard to the gas analyses it will be anticipated that a calculation of the amount of carbon burned at the tuyères and of the carbon absorbed by CO_2 (or "the carbon used in direct reduction") will show figures that will have little significance. Table 25 following shows a calculation of the weight of gas, the weight of blast, and the distribution of carbon between the stack and the tuyères. So long as the amount of carbon blown out as flue dust must be guessed at; and until systematic gas analyses are taken covering appreciable periods of time, no satisfactory figures for these quantities can be given. However, the figures 676 for the ferro furnace, 347 for the spiegel furnace, and 169 for the iron furnace undoubtedly indicate, in a general qualitative way, the pounds of carbon per ton of metal absorbed by CO_2.

TABLE 25.—*Selected data on operation of spiegeleisen blast furnaces.*

Carbon charged per ton of alloy	pounds	3,444
Carbon combined with metal (5.0 per cent C)	do	112
Carbon blown out of stack as dust (9.6 per cent assumed)	do	330
Carbon available as fuel	do	3,002
Carbon from CO_2 in stone	do	260
Total carbon gasified	do	3,262
Gas analysis (by weight):		
CO_2	per cent	11.05
CO	do	30.40
N_2	do	58.08
Carbon in top gas	do	16.07
Weight of top gas per ton of alloy	pounds	20,300
Weight of N_2 in top gas	do	11,810
Weight of air blast	do	15,280
Weight of O_2 in blast	do	3,530
Weight of carbon burned at tuyères	do	2,655
Carbon absorbed by CO_2 or involved in direct reduction	do	347

FUEL REQUIREMENTS.

No relation can be determined between carbon fuel, blast temperature, and slag weight in the spiegel furnace by the method employed for ferromanganese. Table 18 (p. 116) shows that most of the furnaces operating with high-blast temperatures were working on ore mixtures that gave low slag weights, and the furnaces smelting siliceous ores were running with low-blast temperatures. In attempting to determine K_a (the carbon fuel used for average hearth temperatures) as a function of T (the blast temperature) and of V (the slag weight), taking these last as independent variables, it is difficult to distinguish between the effect of changes in blast temperature and of changes in slag weight, as T and V do not vary independently. The method employed below is not so satisfactory as regards accuracy as that used with ferromanganese, but it is simpler and more direct.

The runs in Table 18 were divided into three groups, selected to conform with the following classification:

(1) High-blast temperature and small slag weight.
(2) High-blast temperature and large slag weight.
(3) Low-blast temperature and large slag weight.

The average values of K_a, T, and V for these three groups are given in Table 26 following:

TABLE 26.—*Average values for carbon fuel, blast temperature, and slag weight in the furnace runs represented in Table 18.*

Group.	Runs included.	Furnaces included.	Carbon per ton.	Blast temperature.	Slag weight.
			Lbs.	°*F.*	*Lbs.*
1	43, 46, 48, 49, 55, 57, 62, 64	C, D, H, M, P	2,821	1,221	1,979
2	41, 42, 44, 52, 54, 58, 59, 60, 61, 70	C, E, J, M, R	3,525	1,142	3,349
3	50, 51, 53, 65, 66, 67, 68, 69	E, B, Q, R	4,175	825	4,392

On the assumption that the carbon fuel required for average hearth temperature conditions can be written in the form—

$$K_a=A+BT+CV$$

where A, B, and C are constants, the figures given in Table 26 are sufficient to determine the value of the three constants. Substituting in the above equation three times gives—

$$2{,}821=A+1{,}221B+1{,}979C$$
$$3{,}525=A+1{,}142B+3{,}349C$$
$$4{,}175=A+\phantom{1{,}}825B+4{,}392C$$

From which it follows that—

$$K_a=2{,}365-0.44T+0.49V \qquad (4)$$

As previously stated, K is the quantity of carbon per ton of metal used for any particular run. K_a is the quantity that would be required if the same thermal conditions were maintained in the furnace as in the average furnace. Confusion has often arisen in discussions of blast-furnace fuel economy from identifying the carbon "consumed in" the furnace with the carbon "required by" the furnace. It is true in iron furnace practice that for a given product there is a minimum carbon consumption below which the furnace will not work, but there is practically no upper limit. In making manganese alloys, however, it may be said that there is neither a maximum nor a minimum limit. The actual amount of carbon, K, charged into any furnace is merely the amount the operator believes necessary to maintain a given ratio of coke to ore. In each run, therefore, K, the actual carbon will differ from K_a, the carbon that would on the average be charged in a furnace working with that particular blast temperature and slag weight. $K—K_a$ is a measure of the thermal condition of the hearth. It should be true that when $K—K_a$ is positive the furnace is running hotter than when $K—K_a$ is negative, and this difference in available fuel should influence the amount of MnO left unreduced in the slag.

PERCENTAGE OF MANGANESE IN THE SLAG.

It has been shown for the ferromanganese furnace that M, the percentage of manganese in the slag, is a function of three quantities, namely: B, the ratio of base to silica; $K—K_a$, the excess carbon; and R, the rate of slag formation per square foot of hearth area. The relation between the quantities was found to be—

$$M=21.4-9.16B+1.34R-0.11(K-K_a)$$

In the same way, as is shown below, for the spiegel furnace.

$$M=14.3-5.31B+0.0R-0.09(K-K_a) \qquad (5)$$

This equation is determined as follows: The runs in Table 18, arranged in order of decreasing basicity, are divided into two groups of 15 runs each, group 1 representing runs having the highest basicity and group 2 those having the lowest. The runs are rearranged in order of decreasing $K—K_a$, and two further groups—group 3, representing runs with excess carbon, and group 4, representing runs with deficiency of carbon—are selected. The average values of M B, and $K—K_a$ for these four groups appear in Table 27.

TABLE 27.—*Results of furnace runs arranged to show variation of manganese in slag according to basicity of slag and according to excess or deficiency of carbon fuel.*

Group.	Basicity.	Excess carbon.	Manganese in slag.
		Pounds.	*Per cent.*
1	1.36	123	6.69
2	1.14	137	8.09
3	1.29	292	6.89
4	1.21	331	8.01

From these figures, assuming that the value of M can be expressed in the form given in equation *5*, a simple calculation shows: (*a*) That one unit increase in basicity lowers the manganese in the slag 5.31 per cent; (*b*) That 100 pounds of carbon lowers the manganese in the slag 0.09 per cent.

To show that the rate of driving does not affect the manganese content of the slag, the runs can be again rearranged in order of decreasing R (rate of slag formation per square foot of hearth area), giving two further groups—group 5 representing runs with fast driving, and group 6, representing runs with slow driving. The average values of M, B, $K—K_a$, and R for groups 5 and 6 are given in Table 28.

TABLE 28.—*Results of furnace runs arranged to show variation of manganese in slag according to rate of driving.*

	Group 5.	Group 6.
Basicity ratio	1.24	1.23
Excess carbon, pounds	256	208
Rate of driving, expressed in pounds of slag made per minute per square foot of hearth area	2.86	1.48
Mn in slag, per cent	7.81	7.47

Although the rate of driving differs by 1.38 pounds of slag per minute per square foot of hearth area, the difference of 0.34 per cent manganese in the slag is exactly accounted for by the difference in the values of B and of $K—K_a$. This might well have been anticipated. The curve in figure 10, (p. 127), showing the relation of the percentage of manganese in the slag to the rate of driving for the ferromanganese furnace, is convex downward. When two or more pounds of slag per square foot of hearth area is being produced, the time factor is of importance. The lower end of this curve, however, shows a distinct flattening with a driving rate as low as 1.6 pounds of slag per minute. It is a peculiar coincidence that the average rate of slag formation in both the spiegel practice and in the ferro practice is just 2.1. As regards merely the reduction of MnO, there seems to be no necessity for running the spiegel furnace as slow as the ferro furnace. There is so much more MnO per pound of gross ferro slag to be reduced that the ferro slag should probably enter the hearth only at one-third the rate that is found economical in spiegel practice.

Seemingly, two conclusions may be drawn from these facts:

(1) That overdriving results in a loss of manganese, while underdriving does not result in any corresponding saving of manganese.

(2) That the spiegel furnaces investigated were driven much slower than is shown necessary by any facts presented in this paper.

It is, of course, possible that through other causes furnace difficulties might arise with fast driving, although no evidence of such an effect is at hand.

STACK LOSS.

The "stack loss" is a factor of less importance in the spiegel furnace than in the ferro furnace; the average value is only 6.72 per cent, whereas for ferromanganese it was shown to be 12.8 per cent. The individual values vary from +39.2 to =19.2, a range of 58.4 per cent. The individual values for ferro stack loss varied over a range of 32.9 per cent.

The stack loss shown in Table 18 does not seem to be connected with any of the other operating quantities. The 30 runs were divided in half, giving two groups of 15 runs each; one group included runs with the larger stack losses and the other included runs with the smaller stack losses. The average values of the figures that may have some possible connection are given in Table 29 following:

TABLE 29.—*Results of furnace runs arranged to show comparative effects of large and of small stack losses.*

Item.	Furnaces with large stack loss.	Furnaces with small stack loss.
Stack loss, per cent	15. 5	−2. 22
Silicon in the metal, per cent	1. 92	1. 57
Basicity	1. 25	1. 27
Blast temperature, °F	1, 130	1, 018
Top temperature, °F	633	525
Slag per ton, pounds	2, 930	3, 320
Manganese in slag, per cent	7. 42	7. 69
Carbon burned, pounds	3, 260	3, 665
Recovery, per cent	55. 9	64. 5
Excess carbon, pounds	−35	110

The difference between the stack losses in the two groups is 17.7 per cent. None of the other factors, however, show enough variation to be responsible for such a change in the stack loss. In the first group the silicon content in the metal is higher, as is also the top temperature, while the manganese in the slag is lower. These are changes in the right direction to account for a stack loss, although they can not explain it quantitatively. The fact that the basicity is identical in the two groups is important, as it has been thought by some that "liming up" the slag tends to increase the stack loss.

SUMMARY OF OBSERVATIONS ON SPIEGELEISEN PRACTICE.

The average recovery of manganese in the production of spiegeleisen was found to be 60.22 per cent, the loss of 39.78 per cent being distributed, one-sixth to the stack and five-sixths to the slag.

Although the individual values for stack loss were very irregular, it appears that they were not affected by the basicity of the slag, the temperature of the top gas, nor the carbon fuel charged. There seems to be no way to reduce this anomalous quantity. On the other hand, the evidence does show that the stack loss will not be increased by increasing the basicity of the slag.

The slag loss is decreased by increasing the basicity of the slag. The average basicity of the spiegel slags investigated was $CaO + MgO \div SiO_2 = 1.26$; it seems that this basicity ratio is uneconomically low. The percentage of manganese in the slag is shown to be independent of the rate of slag formation; it is also shown that the spiegel furnaces are operated much more slowly than the manganese oxide reduction demands.

To make a ton of metallic manganese in the form of spiegel requires 78 per cent more carbon fuel than is required to make a ton of metallic manganese in the form of ferro. This greater fuel consumption is due largely to the enormous amount of silica carried in the spiegel ore with the manganese, namely, 1.16 tons of silica to 1 ton of manganese. In this respect the ferro investigation and the spiegel investigation do not overlap. The most siliceous ferro ores used (containing 0.5 ton of SiO_2 per ton of Mn) have a $SiO_2 : Mn$ ratio just equal to that of the least siliceous spiegel ores used. The successful operation of spiegel furnaces on very siliceous ores, however, indicates that the silica specifications on high grade manganese ores have been much too stringent.

Practically all of the conclusions reached in regard to the production of ferromanganese are applicable to spiegel production. In making speigeleisen, the use of better coke and stone has some advantages, but not sufficient to permit such bonuses for low-silica stone and coke as would be permissible in ferro practice; because the silica in the spiegel slag comes largely from the ore gangue, and because manganese in the form of spiegel ore is less expensive than manganese in the form of ferro ore.

Prices for spiegel ores have not been fixed, as for ferro ores, but are subject to individual contract, so a discussion of profits in spiegel operation will not be attempted.

CHAPTER 9.—NATIONAL IMPORTANCE OF ALLOCATING LOW-ASH COKE TO MANGANESE ALLOY FURNACES.

By P. H. ROYSTER.

INTRODUCTORY STATEMENT.

During 1918 an investigation of manganese alloy furnaces with particular reference to the character and quality of the coke being supplied to the furnaces was conducted by the Bureau of Mines. Twelve typical blast furnaces producing ferromanganese and spiegeleisen were examined. These 12 furnaces produced approximately 40 per cent of the total output of manganese alloys in the United States and offer a fair representation of the industry as a whole.

It developed that the poor quality of the coke in general use was responsible for serious waste, both of manganese and of coke. This fact was of vital interest not only to the Bureau of Mines, but also to the Fuel Administration.

FURNACE DATA.

The 12 furnaces studied furnished the following statistics:

Data regarding 12 blast furnaces producing ferromanganese and spiegeleisen.

Item.	Furnaces making ferro-manganese.	Furnaces making spiegeleisen.
Number of furnaces investigated	6	6
Average monthly output of alloys, July and August, tons	9, 966	11, 736
Average coke per ton of alloy, tons	3. 25	2. 12
Total coke consumed, tons	32, 387	24, 880
Average manganese in alloys, tons	73. 0	17. 5
Metallic manganese in alloys, tons	7, 266	2, 100
Conversion loss, per cent	29	38
Metallic manganese in ores, tons	10, 266	3, 380
Average manganese in ores, per cent	40. 6	12. 8
Total ore consumed, tons	25, 276	26, 502

IMPORTANCE OF ALLOCATING LOW-ASH COKE TO MANGANESE FURNACES.

A total of about 57,000 tons of coke per month was being consumed by the 12 furnaces under review. This coke was found to contain an average of nearly 14.5 per cent ash, or over 7 per cent silica.

The principal losses of manganese in the manufacture of alloy are in the slag, these losses being roughly proportional to the slag volume. The controlling slag-forming element is silica. Silica in coke is more harmful than silica in ore because it not only increases the slag volume but also diminishes the efficiency of the coke, making more coke necessary, which in turn introduces more silica, thereby again increasing the slag volume and the coke requirement. In fact, if the coke contains as much as 20 per cent ash, the vicious circle may be complete, all further additions of coke serving no purpose whatever. It is obvious that the more nearly this extreme is reached, the more wasteful the process becomes, both as to manganese and as to coke. Conversely, with better coke, less coke is needed and more manganese is saved.

Comparative observations made and recorded demonstrate if the 12 furnaces under review had been supplied with coke containing 8 per cent ash instead of 14.5 per cent, the same monthly tonnage of alloys could have been produced with about 40,000 tons of coke instead of the 57,000 tons actually used. At the same time, the conversion loss of metallic manganese would have been reduced from 29 per cent to 14 per cent for ferro, and from 38 per cent to 29 per cent for spiegel. The net result of this last would have been a saving of 1,826 tons of metallic manganese, equivalent to 4,560 tons of 40 per cent ore in the case of ferro; and a saving of 420 tons of metallic manganese, equivalent to 3,230 tons of 13 per cent ore in the case of spiegel.

Extending these figures, which represent actual conditions as to about 40 per cent of the country's (1918) output of manganese alloys, to cover the entire estimated requirement of such alloys for the year 1919, gives the following results:

1. Estimated requirement for 1919, 280,000 tons metallic manganese of which 20 per cent shall be in the form of spiegel.

	Tons of coke.
2. 56,000÷17 per cent=330,000 tons spiegel×2.12=	699,600
224,000÷72 per cent=312,000 tons ferro×3.25=	1,014,000
Total requirement of 14.5 per cent ash coke	1,713,600
Saving through use of good quality coke, 31 per cent	532,000
Total requirement of 8 per cent ash coke	1,181,600

	Poor coke.	Good coke.	Saving.
3. Spiegel conversion loss, per cent	38	29	....
Tons Mn needed for spiegel	90,000	79,000	11,000
Tons 13 per cent ore needed for spiegel	690,000	606,000	84,000
Ferro conversion loss, per cent	29	14	....
Tons Mn needed for ferro	315,000	262,000	53,000
Tons 40 per cent ore needed for ferro	786,000	655,000	131,000

In short, by allocating low-ash coke during 1919 to the manganese-alloy furnaces, there may be saved over half a million tons of coke; about 131,000 tons of manganese ore of ferro grade; about 84,000 tons of ore of spiegel grade; and a total of about 64,000 tons of metallic manganese.

The effect of such allocation on the production of pig iron must, however, be considered. The pig-iron industry would be deprived of roughly 1,200,000 tons of good coke and receive instead about 1,700,000 tons of relatively poor coke.

In the manufacture of pig iron, poor coke does not cause any increase in loss of metallic iron. The substitution of 14.5 per cent ash coke for 8 per cent ash coke would, however, cause about 10 per cent decrease in production and about 20 per cent increase in the use of coke. The decrease in production is the direct result of having to use an increased amount of coke.

The net effect would be a decrease in the production of pig iron of about 120,000 tons, and an increase in coke consumption of about 240,000 tons. One hundred and twenty thousand tons is approximately one-third of 1 per cent of the total pig-iron output of the country.

However, as the pig-iron industry would receive 500,000 tons more coke than it would be asked to release there would be enough coke left, after supplying the above additional requirement of 240,000 tons, to make up the deficiency in pig iron and still leave a margin of 120,000 tons.

Therefore the net coke supply would be actually increased, while maintaining the pig output at its present level; but about 130,000 tons additional furnace capacity would have to be provided.

The greater part of this, namely, at least 100,000 tons,[a] would be furnished by the manganese-alloy industry, because its furnace requirements would be reduced by this amount, while producing the same tonnage of alloys, on account of having high-grade coke. The small balance must be made up from some other source. As bearing upon this point, it is interesting to note the following statistics:

Data on pig-iron blast furnaces in the United States.

	Number.
In blast, Aug. 31, 1918	371
In blast, Sept. 30, 1918	364
Out of blast, Sept. 30, 1918	74
Total furnaces, Sept. 30, 1918	438
Average yearly capacity per furnace (approximate), tons	100,000

CONCLUSION.

Without doubt, a large proportion of the furnaces idle (Oct. 1, 1918) are small and poorly equipped. But, in view of the immense national importance of allocating low-ash coke to the manganese-alloy furnaces, it would seem justifiable to exert pressure on the pig-iron industry to make up the resulting slight loss of furnace capacity from among the furnaces now idle.

The figures given herein are based on a grade of coke which is to-day admittedly scarce; on the other hand, the total requirements of the manganese-alloy furnaces are only a small proportion of the total coke output of the country. The best coke available, whatever its grade may be, should be allocated to these furnaces; and if the average ash proves to be somwhat more than 8 per cent, the savings made, while proportionately less than shown above, will still be of the utmost importance.

In making these figures, commercial aspects have been constantly kept in mind. The figures are not theoretical; they represent what it is believed can actually be accomplished.

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[a] Furnaces capable of making 68,000 tons of spiegel per year could be released for pig production. The estimate of 100,000 tons pig from these furnaces is low. A 100-ton pig furnace will not make 68 tons of spiegel.

CHAPTER 10.—ELECTRIC SMELTING OF DOMESTIC MANGANESE ORES.

By H. W. Gillett and C. E. Williams.

INTRODUCTORY STATEMENT.

The utilization of domestic manganese ores has been an important war-time problem, and its bearing on national self-sufficiency makes it worthy of attention even in times of peace. This paper deals only with the utilization of low-grade domestic ores by electric smelting.

As a rule, blast furnaces can not, in normal times, economically use ores averaging much below 40 per cent Mn and much above 12 per cent SiO_2, although during the war these limits were exceeded. The United States contains large tonnages of ores lower in Mn and higher in SiO_2 which may be amenable to electric smelting. An electric-furnace plant can be constructed more rapidly and can be economically operated in smaller units than a blast furnace; also it can be situated wherever a suitable supply of ore and a suitable water-power development are found near each other. An ore deposit inadequate to supply a modern blast furnace would last a rather large electric furnace for a considerable time. Hence, the smaller electric unit can be so located that the transportation charges are less than if the ore were shipped to a more distant blast furnace.

The writers wish to acknowledge the valuable assistance of Messrs. E. L. Mack and F. C. Ryan, of the Bureau of Mines, and of Lieuts. L. S. Deitz, jr., and J. G. Thompson, Ordnance Department, and Pvt. E. O. Denzler, of the Chemical Warfare Service, who aided in the experimental and analytical work. Grateful acknowledgment is also made to the Chemical Department of Cornell University for the use of its electric-furnace equipment and of its analytical laboratory.

PRESENT DEVELOPMENT OF ELECTRIC SMELTING OF DOMESTIC ORES.

Electric smelting of local ores has been considerably developed. The Southern Manganese Corporation, Anniston, Ala., has eight furnaces, some of them of 3,000 kilowatt size, operating on local ore.[a] The Anaconda Copper Co.[b] will soon be operating five 3,000-

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[a] Swann, T., (news item): Met. and Chem. Eng., vol. 18, 1918, p. 512.

[b] Anonymous, Electric furnace for manganese ores: West. Elec., vol 73, 1918, p. 218; Chem. Abs., vol 12, 1918, p. 1856.

kilowatt furnaces at Great Falls, Mont., on uncalcined rhodochrosite ore from Butte, Mont., and will probably also use some ore from Philipsburg, Mont. Washed Philipsburg ore is also smelted by the Bilrowe Alloys Co., Tacoma, Wash., in six 350 to 400-kilowatt furnaces, and this firm has also smelted ore from the Olympic Mountains in Washington.

The Noble Electric Steel Co., Heroult, Calif., smelts California ores in two 1,300 to 1,500-kilowatt furnaces. The Pacific Electro Metals Co. makes ferromanganese and silicomanganese in a 3,000-kilowatt furnace at Bay Point, Calif., from California ores.

The Iron Mountain Alloy Co., Utah Junction, Calif., runs a 1,200-kilowatt and an 1,800-kilowatt furnace on Colorado, Utah, and Nevada ores.

The Western Reduction Co., Portland, Oreg., is just starting the production of ferromanganese in a 700-kilowatt furnace.

ELECTRIC SMELTING PRACTICE ON FERROMANGANESE.

FURNACES USED.

Practice in making ferromanganese differs slightly at the various plants, but in general is uniform. Keeney[a] has recently described the Iron Mountain Alloy Co.'s practice. Furnace voltages (electrode to charge) run from 55 volts in the 350-kilowatt size to 85 in the 3,000-kilowatt size. The furnaces are usually three-phase, three-electrode, open-top furnaces, the exceptions being those of the Bilrowe Alloys Co., which are single-phase, and those of the Noble Electric Steel Co., which are of a three-phase, four-electrode type. The general type of furnace used is the rectangular form described by Lyon, Keeney, and Cullen.[b] The Noble furnace is described by Vom Baur.[c]

The furnace lining is usually carbon or magnesite, rarely water-cooled fire brick. The furnaces are charged continuously and tapped every two hours. Slag and metal are tapped together into a settler. A few plants use round electrodes jointed for continuous feed, which should reduce the electrode consumption. However, in most plants the feed is not continuous, and the butts are scrapped. Seemingly, a Scott-connected, two-phase circuit from a three-phase circuit with the conducting bottom of the furnace as the neutral electrode, and two upper electrodes, might reduce the electrode consumption by presenting only two instead of three electrodes to oxidation, while still allowing the construction of a furnace of high kilowatt capacity without the upper electrodes being required to carry too much current.

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[a] Keeney, R. M., The manufacture of ferro-alloys in the electric furnace. Bull. Am. Inst. Min. Eng., vol. 140, August, 1918, pp. 1321-1373.

[b] Lyon, D. A., Keeney, R. M., and Cullen, J. F. The electric furnace in metallurgical work: Bull. 77, Bureau of Mines, 1914, p. 113.

[c] Rodenhauser, W., Schoenawa, J., and Vom Baur, C. H., Electric furnaces in the iron and steel industry, 1917 ed., p. 365.

RAW MATERIALS EMPLOYED.

The ores of mixtures used at all these plants vary between 39.5 to 43.5 per cent manganese and 15 to 25 per cent silica with low phosphorus, averaging about 40 per cent manganese, 20 per cent silica, 0.20 per cent phosphorus, and less than 4 per cent ferro. The Great Falls, Mont., plant will be an exception, as it will operate largely on Butte rhodochrosite ore which is extremely low in silica and phosphorus.

The plants all flux the ore with limestone, except the plant at Heroult, Calif., which uses a little fluorspar. Also they produce a slag whose average composition is 10 to 12 per cent Mn., 35 to 40 per cent $CaO + MgO$, about 30 per cent SiO_2, and such small amounts of Al_2O_3 as are carried by the ores. The latter are all low in alumina with the exception of occasional lots smelted at Anniston, Ala.

AVERAGE RESULTS OBTAINED.

The power consumption and the electrode consumption in the various furnaces, brought to the basis of a 3,000-kilowatt furnace, average 5,000 kilowatt-hours and 170 pounds of carbon electrodes per long ton of 80 per cent ferromanganese.

As the ores are low in iron, a little iron ore, or better and more generally used, some steel scrap, is added to supply enough iron to bring the alloy to 80 per cent manganese. The alloys contain 0.20 to 0.40 per cent phosphorus, the former figure being aimed at, but the latter being taken by the trade while the ferro supply is low. The California ores are low in phosphorus; consequently the California alloys meet the lower figure. The silicon content runs under 2 per cent, though it sometimes goes up to 5 per cent in the Anniston product when ores higher than usual in silica are employed.

The Anniston, Anaconda, and Iron Mountain plants use anthracite coal as reducer; the Bay Point and Tacoma plants use coke; and the Heroult plant uses about half coke and half charcoal.

The recovery of manganese averages 75 per cent, being nearer 80 per cent on ores of 15 per cent silica and nearer 70 on those of 25 per cent silica content. From a sixth to a third of the conversion loss is due to volatilization and dusting, and slag losses account for the rest. In the blast furnace operating on low silica ores these losses are exactly reversed, slag accounting for only a sixth to a third and the rest being due to volatilization and dusting.[a]

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[a] Newton, Edmund, Manganiferous iron ores of the Cuyuna District, Minn., Bull. 5, Minn School of Mines Sta., University of Minn., 1918, p. 74.

EFFECT OF HIGH SILICA IN ORES.

It is seen that the electric smelting plants use ores higher in silica than are standard for blast-furnace practice, but that 25 per cent silica is the present limit in ores used for making ferromanganese.

The trouble that silica causes is reflected in the manganese price scale of the War Industries Board,[a] by which an ore with 40 per cent manganese and 8 per cent silica is worth $40.80 per ton and one with 35 per cent manganese and 35 per cent silica only, $7.45 per ton. The scale does not run below 35 per cent manganese, but if it did an ore with 27 per cent manganese and 35 per cent silica would be worth nothing, the penalty for silica entirely wiping out the value of the manganese even though calculated at the same price per unit as given in the scale for a 35 per cent manganese content.

CONCENTRATION OF ORES.

The Philipsburg and the Butte, Mont., districts contain vast tonnages of manganese ores containing around 25 per cent manganese and 35 per cent silica, to say nothing of the rhodonite ores of Colorado and of the tailings from concentrating ores by washing or screening.[b]

Some concentration of domestic ores is done,[c] but in general concentration, is not very successful[d] and a slight improvement in the method of handling low-grade ores might make it more desirable to smelt the run-of-mine ore than to attempt concentration. It is stated[e] that some ores could be concentrated after fine crushing; also that leaching and electrolytic deposition of the manganese oxide is possible. The product would, however, be fine powders and would be extremely difficult to smelt in the usual fashion, keeping the shaft full of charge, unless first briquetted or sintered. It is not certain that briquetting or sintering could be successfully applied.

REDUCTION OF SILICON.

Even though the usual basic slag be employed when smelting ores high in silica, silicon is reduced and enters the alloy when the slag volume is high. Newton[f] states that this is true in the blast furnace. Swann[g] also indicates it for electric-furnace operation, stating that slags from electric smelting containing 12 to 14 per cent Mn, 40 per cent CaO, and 30 per cent SiO_2 can be smelted to an alloy containing 60 per cent Mn and 19 per cent Si. Had the original smelting been carried on far enough to reduce more of the manganese, considerable silicon would have been reduced also. A high temperature and a large amount of reducing agent are needed to drive the manganese out of the slag, and these conditions favor the reduction of silicon.

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[a] Chapter 1 of this bulletin.

[b] See Pardee, J. T., Manganese at Butte, Mont.: U. S. Geol. Survey Bull. 690, 1918, pp. 111-130; Harder, E. C., Manganiferous iron ores; U. S. Geol. Survey Bull. 666-ee, 1917, 13 pp.

[c] See Swann, T., place cited; Anonymous, Manganese concentrator at Philipsburg, Mont.: Met. and Chem. Eng., vol. 18, 1918, p. 625.

[d] See Pardee, J. T., work cited, p. 11; Harder, E. C., work cited, p. 13; Newton, E., work cited, p. 57.

[e] News item, Manganese in California: Chem. and Met. Eng., vol. 19, 1918, p. 702.

[f] Newton, E., work cited, p. 72.

[g] Swann, T., place cited.

As in smelting high-silica ores some silicon will in any event go into the alloy; it appears that it will probably be best to make an alloy with 15 to 20 per cent silicon. In fact, utilization of ores high in silica appears to demand such a silicon content.

PRODUCTION OF SILICOMANGANESE.

Very little silicomanganese has been used in the United States, although it is widely used abroad, being made in large amounts in Sweden, France, and Switzerland. Harden [a] gives the Swedish production of ferromanganese as 2,000 tons per year and that of silicomanganese as 3,500 to 4,000 tons. He says the silicomanganese is made from a mixture of manganese ore and quartz. Lyon, Keeney, and Cullen [b] state that it is also made from rhodonite ore. Swedish practice is said [c] to utilize Bessemer and blast-furnace slags for the manufacture of silicomanganese.

While the demand for silicomanganese is still low in the United States, one firm, the Pacific Electro Metals Co., is making it. The ore used contains 40 per cent Mn, 23 per cent SiO, very little phosphorus, and practically no CaO, MgO, or Al_2O_3. Quartz is added to the ore to form a mixture of about 30 per cent Mn and 35 to 40 per cent SiO_2. A low-grade ore of this composition would be acceptable. Steel scrap is also added to the charge, and the alloy produced runs 50 to 55 per cent Mn. 20 to 25 per cent Si, 0.6 per cent C, 0.2 per cent P, and the remainder iron. The power consumption per long ton of alloy produced is said to be 5,800 kw.-hours, and the coke consumption 1,400 to 2,000 pounds in a 3,000-kw. furnace. The furnace is run without flux, low-ash coke being desirable, and the whole charge is reduced to metal, just as in ferrosilicon smelting. This would be possible only with a practically gangue-free ore.

The absence of slag avoids the usual loss of manganese in the slag incident to making ferro-alloy—the manganese recovery in making silicomanganese being 95 per cent. That is, with an ore of this grade, 20 per cent more manganese is recovered in alloy than when ferro-alloy is made—or 4 tons of ore made into silicomanganese gives as much usable manganese as 5 tons made into ferromanganese.

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[a] Harden, J., Utilization of manganese ores in Sweden: Met. and Chem. Eng., vol. 17, 1917, p. 701.

[b] Lyon, D. A., Keeney, R. M., and Cullen, J. F., work cited.

[c] News item, Manganese from steel slags: Min. and Sci. Press, vol. 116, Apr. 6, 1918, p. 486.

As regards conservation, silicomanganese, rather than ferro, should therefore be made from ores low in lime and alumina.

The addition of iron is probably due to the buyer's specification. It is illogical to add iron, use power to melt it, and then have to ship it to the steel works where all it does is to chill the steel bath when the alloy is added. By leaving out the iron an alloy containing 68 per cent Mn, 2 per cent Fe, 29.5 per cent Si, 0.3 per cent C, and 0.23 per cent P could be produced and should be preferable to the alloy now made.

FUTURE OF HIGH-SILICA ORES DEPENDENT ON USE OF SILICOMANGANESE.

Any utilization of low-grade ores high in silica postulates that the steel industry can and will use silicomanganese.

Only a few brief calculations are required to prove that there is enough steel produced for castings, forgings, etc., to which both ferromanganese and ferrosilicon are added, to absorb a very large production of silicomanganese. Common sense and foreign practice, as well as actual large-scale experiments in this country, indicate not only that this alloy can be used, but that its use is distinctly desirable.[a]

It should be noted that the high-silicon alloys contain very little carbon, so that where recarburization is now done by ferromanganese or spiegel, other means of recarburization, such as the use of charcoal, coal, or coke, must be resorted to when silicomanganese is substituted for ferro or spiegel. The low carbon content would, on the other hand, often be a distinct advantage.

CARBON CONTENT OF MANGANESE ALLOYS.

The relation of the silicon to the carbon content in alloys containing 60 to 80 per cent manganese is shown by the curve in figure 13. Silicospiegels of lower manganese content will give a curve lying below this one, as iron takes up less carbon than manganese. The smelting temperature, as well as the practice with regard to carbon, affects the carbon content; but for alloys made by electric smelting at normal smelting temperature and with an excess of carbon, the curve is closely accurate. As regards alloys with less than 5 per cent silicon, the carbon content may vary about 0.75 per cent from the values shown by the curve. For alloys with 10 per cent silicon the deviation will be less than 0.5 per cent carbon; for those with 15 per cent silicon and higher, the deviation will be only 0.1 to 0.2 per cent carbon.

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[a] See Swann, T., The development of the ferromanganese industry in the United States since 1914: Chem. and Met. Eng., vol. 19, 1918, p. 672.

REASONS FOR EXPERIMENTAL WORK.

It is natural that commercial electric-furnace plants should procure the highest grade raw material available. For this reason little is known about the actual working possibilities of low-grade manganiferous ores. If any experiments have been made on such ores, the records have not been published. Their behavior in the electric furnace can, to a certain extent, be predicted by a metallurgist skilled in smelting high-grade ores, but beyond a certain point, such predictions become little more than speculation.

As the United States has immense known deposits of low-grade ores, many of which are not amenable to ordinary methods of gravity concentration, the study of the smelting of such materials in the electric furnace, as herein described, was undertaken by the Bureau of Mines. It was felt that if information could thereby be secured that would make any portion of these immense ore reserves commercially available, a real contribution would be made to the country's resources.

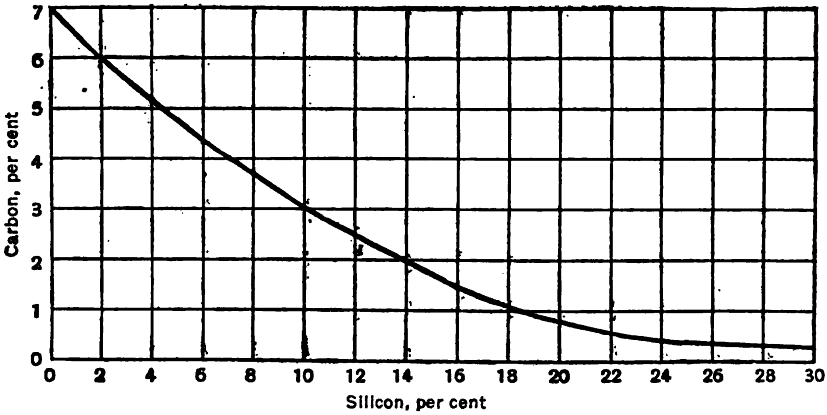

FIGURE 13.—Curve showing relation of carbon to silicon in ferro-silicomanganese alloys.

CLASSES OF LOW-GRADE ORES.

In the utilization of low-grade domestic ores, one would naturally start with the best of such ores—that is, those highest in manganese or lowest in silica, leaving the leaner ores until experience had been gained on the better ores. Attention should, then, be paid (1) to ores high in silica, low in manganese, and low in phosphorus, like the low-grade Butte ores; (2) to similar ores, but high in ratio of phosphorus to manganese, such as the Philipsburg ores; and (3) to those high in iron, such as the Cuyuna ores.

Exact analyses of the low-grade Butte ores are not available, but from Pardee's statements[a] we may assume that large tonnages of ore containing 25 per cent Mn, 40 per cent SiO, 3. 5 per cent Fe, approximately 5 per cent Al_2O_3, less than 0. 05 per cent P, 13 per cent loss on ignition, and only a trace of CaO, should be available. Obviously, such ore would be eminently fitted for the manufacture of silicomanganese, smelting without a slag, provided the alumina did not form a stiff slag of sufficient volume to give trouble. In attempting complete reduction of this ore some aluminium would probably be reduced into the alloy; but as this aluminium would act first in the deoxidizing work of the alloy in steel, being thus eliminated and at the same time protecting the manganese, its presence would probably not be detrimental. In fact silicomanganese-aluminum of high aluminium content is used abroad as a deoxidizer. The low phosphorus content of this ore and its low content of slag-forming constituents make it appear one of the most promising of the low-grade ore.

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[a] Pardee, J. T., Manganese at Butte, Mont.: U. S. Geol. Survey Bull. 690, 1918, pp. 123-125.

The Philipsburg (Montana) ore is low in iron and the deposits are large. Special lots of low-grade Philipsburg ore have given the following analyses:

Results of analyses of low-grade Philipsburg (Montana), manganese ores.

Lot No.	Mn.	Fe.	SiO.	CaO-MgO-BaO.	Al_2O_3.	P.	Loss on ignition.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
1	29.5	2.5	25.0	9.6	1.5	0.105	18.7
2[a]	26.6	2.8	35.9	6.7	5.8	.24	13.3
2b[b]	24.1	1.6	37.7	7.7	1.6	.37	12.7
2c[c]	28.7	2.4	28.5	4.1	3.4	.14	14.6
4[c]	26.4	2.5	36.4	7.5	1.9	.21	11.8

[a] Contained both lump and screenings. [b] Lump picked off grizzly. [c] Screenings, ⅛-inch screen.

The lump and screenings are obtained in the concentration (by hand picking and screening only) of the run-of-mine ore. The screenings are thought to constitute about 20 per cent of the ore as mined. This type of ore is too basic to allow running without a flux. The ratio of phosphorus to manganese content is high. There would then be two problems, namely, how to flux the ore and how to obtain an alloy low in phosphorus.

EXPERIMENTAL WORK.

To solve these and similar problems, advantage was taken of the electric-furnace equipment of the Bureau of Mines field office at Cornell University, Ithaca, N. Y. Small-scale experiments of this sort are admittedly inconclusive. They go a long way, however, toward replacing speculations and proving theories, and may safely be used as guides to large tests on a commercial scale.

The ores used in the tests were obtained from Philipsburg. Their analyses are given in the foregoing table.

DESCRIPTION OF FURNACE USED.

A single-phase furnace with conducting bottom, with a carbofrax (carborundum) lined shaft 14 inches in diameter and 18 inches deep, surmounted by a fire-brick shaft 12 inches deep, was built. The upper electrode was a square carbon 4 by 4 inches. The carborundum lining worked well, and required few repairs. What little patching was required was readily done with a cement of fine carborundum and water glass. Inasmuch as many of the slags were acid, a magnesite lining would not have been satisfactory. As the runs were intermittent a carbon lining would have been liable to oxidation while the furnace was cooling between runs. The furnace uses about 65 kw., or 1,450 amperes at 50 volts, with an average power factor of 90.

PRELIMINARY TESTS.

Some continuous runs of about 14 hours were made with screenings that passed a ½-inch screen, keeping the shaft full of ore and tapping at intervals. The ore contained much fine dust, and was too fine to bridge well; hence in most runs the furnace was run discontinuously, one charge being smelted and tapped before a succeeding charge was added. The tap hole was closed with a charcoal plug and a wad of fire clay rammed over the plug. In tapping, the fire clay was dug out, when the charcoal could be readily barred out. Starting with a cold furnace, it took about 8 hours to smelt three batches of ore each of 100 pounds. To obtain a slag low in manganese, and a high recovery of manganese in the alloy, it was necessary to run the furnace hot, using not less than 60 to 65 kw. A fairly fluid slag and a considerable excess of reducer were also required. The excess of reducer does not usually come out of the furnace with the slag, but accumulates from heat to heat in the furnace bottom, so that in commercial operation after the first few heats considerably less reducer would be needed than was required in the experimental work. The excess of coke or charcoal holds some slag and metal in its pores so that the "dross" scraped out of the furnace after a series of runs is always higher in manganese than the slag. In commercial operation the "dross" would remain in the furnace and the metal lost in operation on an experimental scale would be recovered.

The manganese content of the slag on the first heat in a cold furnace is always higher than on subsequent heats, even though the slags may be tapped at equal temperatures.

STANDARDIZATION OF FURNACE.

For these reasons the manganese recovery to be expected in the commercial operation would be higher than that obtained in small-scale experiments. Power consumption and electrode consumption in a small furnace, when the furnace is started cold and only a short run is made will be much higher than in commercial practice. A rough factor for calculating power consumption in large-scale operations can be obtained by running a high-grade ore in the small furnace and comparing the power used with that required in commercial operation on such an ore.

Such a standardization was made with a high-grade ore—Philipsburg concentrates—analyzing 47.7 per cent Mn, 1.9 per cent Fe, 8.7 per cent SiO_2, 2.1 per cent CaO, 1.1 per cent Al_2O_3, 0.08 per cent P, 14.2 per cent loss on ignition. The usual basic slag was made. The results follow:

Run 52.—Three charges, each consisting of 100 pounds of ore, 3 pounds of steel scrap, 14 pounds of coke (metallurgical coke, ¼-inch mesh; analyzing 4.75 per cent H_2O, 10.3 per cent ash, of which 1.4 per cent was Fe, 4.3 per cent SiO_2, 2.5 per cent Al_2O_3, 0.8 per cent CaO), 14 pounds of charcoal (crushed to about ½ inch), and 20 pounds of $CaCO_3$ (marble, crushed to ½ inch). Time of run, 8 hours 10 minutes; power used, 440 kw.-hours (180 kw.-hours, first heat; 140, second heat; 120, third heat). Obtained 158.2 pounds of metal, containing 82.0 per cent Mn, 2.0 per cent Si, 0.12 per cent P; 84.3 pounds of slag, containing 8.5 per cent Mn; and 11 pounds of dross containing 34.5 per cent Mn. Mn distribution: In metal 90.5 per cent, in slag 4.2 per cent, in dross 2.7 per cent, volatilization and dusting loss 2.6 per cent. Recovery of P in metal, 79 per cent.

As, in commercial practice, not over 4,000 kw.-hours per long ton of alloy would be required on such an ore in a 3,000-kw. furnace, or 2.2 kw.-hours per pound of metallic manganese in the alloy, and as the last two taps in the small furnace used 3.0 kw.-hours per pound of manganese in the alloy, it appears that 75 per cent of the power used per pound of alloy in the experimental furnace on the last two taps of a run on 300 pounds should give the approximate power required commercially. A previous run (No. 50) on this high-grade ore, using the same flux, and adding 5 pounds of mill scale, to supply iron, 10 pounds of burnt lime, and 33 pounds of charcoal per 100 pounds of ore, gave an alloy of 77 per cent Mn, 2 per cent Si, 0.16 per cent P, with a manganese recovery of 79 per cent. Some of the heats were not given enough power, and some of the slags were high in manganese. It was calculated that under correct operating conditions 88 per cent of the manganese would have been recovered. This calculation was checked by the results of run 52. The results of these runs indicated that a mixture of half coke and half charcoal was as good a reducer as all charcoal.

FLUXING TESTS WITH HIGH-SILICA ORE.

A series of fluxing tests was made with a high-silica ore, namely, the Philipsburg screenings, lot 4 (see table of analyses), containing 26.4 per cent Mn, 2.5 per cent Fe, 36.4 per cent SiO_2, 7.5 per cent CaO+MgO, 1.9 per cent Al_2O_3, and 0.21 per cent P, with 11.8 per cent loss on ignition. In all the runs 10 pounds of coke and 10 pounds of charcoal were used per 100 pounds of ore, except in run 73 in which 15 pounds of each were used. No steel scrap was added except 0.5 pound per 100 pounds of ore in run 70. The results of the runs are summarized as follows:

Results of fluxing tests with high-silica ore.

CHARGES AND POWER CONSUMPTION.

Run No.	CaO per 100 pounds of ore.	CaF_2 per 100 pounds of ore.	CaO per pound of silica, including CaO equivalent to bases in ore.	CaF_2 per pound of silica.	Ore used.	Time of run.	Power consumed.
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*H. min.*	*k.w.-hours.*
70	72	....	1.30	....	300	8 20	522
69	28	....	.60	....	300	7 20	450
67	19	7	.36	0.19	300	7 25	453
68	....	14.5	.20	0.40	300	7 20	452
73	None till end, then 3.5.	10	None till end, then 0.30.	0.28	200	7 8	469

PRODUCTS AND POWER CONSUMPTION.

Run No.	Alloy made per 100 pounds.	Analysis of alloys.			Slag made per 100 pounds.	Mn in slag.	P in slag.	Kw.-hours per pound of alloy, including all heats.	Kw.-hours per pound of Mn, including all heats.
		Mn.	Si.	P.					
	Pounds.	*Per cent.*	*Per cent.*	*Per cent.*	*Pounds.*	*Per cent.*	*Per cent.*		
70	23.8	71.0	13.0	0.60	93	6.0	0.04	7.3	10.2
69	18.4	71.4	11.6	.73	61	11.5	.06	8.2	11.4
67	21.9	73.0	13.0	.68	65	14.0	.04	6.9	9.5
68	26.3	71.0	15.5	.60	59	12.0	.04	5.75	8.1
73	26.7	72.7	19.8	.57	17	23.1	.04	8.8	12.0

DISTRIBUTION OF ELEMENTS IN PERCENTAGES OF TOTAL.

Run No.	In metal.			In slag.		In dross.		Loss.	
	Mn.	Si.	P.	Mn.	P.	Mn.	P.	Mn.	P.
70	63	8½	68	21	18	....	....	16	13
69	50	6	64	26	18	7	5½	17	12½
67	60	8	70	34	3½	....	(a)	3	13½
68	71	11	75	27	11½	....	(a)	2	13½
73	73	14½	72½	21	5	6	10½	0	12

a Amount of dross small; included in weight of slag and in sample of slag for analysis.

More power was used in run 70 than in the others because of the large amounts of $CaCO_3$ charged. The slag was very stiff in run 69 and an arc had to be used to tap. More power might have given a slightly better recovery.

In run 73 more reducer was used than in the other runs, and no flux was added till the end. Two batches of 100 pounds of ore each were added before tapping. The metal tapped, but the slag was too stiff to run; the fluxes were than added, and 45 kw.-hours were used (included in the 469 total in the table) to get the slag out. It then tapped well.

With the same power input for runs 69 and 67 as for run 70, run 67 would have given better results than run 70, but the results of run 69 would not have been as satisfactory as for run 70, the slag being too stiff. The acid slags containing fluorspar, however, tapped well.

Considering the cost of power and fluxes, it appears that the acid slags of runs 67 and 68 were preferable to the basic slag of run 70. It is noteworthy that the basic slag did not prevent the reduction of silicon in run 70. The recovery of manganese was higher in run 73, operating without flux until the furnace got clogged with slag and then fluxing out the slag, but the power consumption was too high. The flux of run 68—all CaF_2, except for the CaO in the ore itself—gave the best results metallurgically. The slag of run 68 was very fluid, and some of the CaF_2 could probably have been replaced by CaO, using a flux intermediate between those of runs 68 and 73. Slags made with fluorspar, are, for an equal Ca content, much more fluid than those made with lime.

ADVANTAGES OF ACID SLAG.

The percentage of manganese is somewhat lower in a strongly basic slag than in an acid slag, but the weight of slag formed is much greater. Hence, the weight of manganese lost in the slag is greater with the basic slag, when the SiO_2 content of the ore is much above 25 per cent. When the ore is fluxed to a basic slag the cost of handling and disposing of the excess slag formed must be considered. The cost of the lime flux for the basic slag would, of course, be lower than that of the fluorspar required to produce an acid slag fluid enough to tap; although in a large furnace, where the tap hole would not freeze as readily as in the small one when tapping a sticky slag, the amount of fluorspar could doubtless be materially reduced.
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It will be noted that in the best run (No. 68), the ratio of phosphorus to manganese in the alloy was 1:118, whereas in the ore the ratio was 1:125. Thus it appears that phosphorus and manganese were recovered in about the same proportion.

TEST OF REDUCERS.

On account of the variation in commercial practice as to the reducer used, anthracite, coke, or a mixture of coke and charcoal being used by different firms, a series of tests was made with reducers.

Some preliminary runs in a small furnace taking only about 20 pounds of ore per charge were made. The Philipsburg screenings used contained 29.5 per cent Mn, 2.5 per cent Fe, 25.0 per cent SiO_2, 9.5 per cent $CaO + MgO$, 1.5 per cent Al_2O_3, and 0.105 per cent P, and showed 18.7 per cent loss on ignition. The following results were obtained, the same power input being used in each run:

Results of tests of reducers in small electric furnace.

	Run 20.	Run 21.	Run 22.
Charge:			
Ore .... pounds..	20	20	20
Coke .... do....	6	.....	.....
Anthracite .... do....	.....	5	.....
Charcoal .... do....	.....	.....	5
Calcium fluoride (CaF_2) .... do....	2½	2½	2½
Analysis of alloy made:			
Manganese (Mn) .... per cent..	74.0	77.3	75.3
Silicon (Si) .... do....	13.0	4.6	12.5
Phosphorus (P) .... do....	0.24	0.32	0.20
Recovery in alloy:			
Manganese .... do....	66	43	66
Phosphorus .... do....	57	52	40

Anthracite gave the lowest recovery. The recovery with coke was the same as that with charcoal.

Runs 48 and 49 were then made in the larger furnace. The slag used was calculated for a ratio of CaO to SiO_2 of 0.4:1, which in the later fluxing tests was shown to be too stiff a slag for good recovery. Seventy-five pounds of ore was used per charge and the furnace was run 12 to 13 hours. The data on the average ore mixture used, the composition of the charge, and the results follow:

Data on reducer tests with large electric furnace.

	Run 48.	Run 49.
Ore mixture:		
Mn .... per cent..	28.7	29.0
Fe .... do....	2.4	2.2
SiO_2 .... do....	28.5	26.5
CaO .... do....	4.1	4.2
Al_2O_3 .... do....	3.3	2.7
P .... do....	.14	.16
Loss on ignition .... do....	14.5	16.5

Charge (weight of constituent to each 100 pounds of ore):		Run 48.	Run 49.
$CaCo_3$	pounds	12	12
Charcoal	do	12.3	None.
Coke	do	None.	13.5
Quantity of ore charged	do	900	825
Time of run	hours	12.25	13
Power used	kw.-hours	714	710
Weight of alloy made	pounds	182.0	146.1
Composition of alloy:			
Mn	per cent	74.0	76.0
Si	do	10.0	8.0
P	do	.34	.38
Weight of slag made	pounds	496	462
Manganese in slag	per cent	16.4	18.3
Recovery in alloy:			
Mn	do	52	46
P	do	49	42

As regards the two tests, although the ore used in run 49 was slightly higher in manganese and lower in silica, and although more power was used per unit weight of ore, charcoal gave better results than coke. However, neither result was satisfactory, the slag being too acid and holding up too much manganese. A slag of the same acidity made with fluorspar would have shown better results. The amount of reducer in both tests was too small for good results.

COMMENTS ON USE OF RHODOCHROSITE.

As the lot of ore used in run 48 was exhausted before run 49 was finished, the last few charges in run 49 were made with a mixture of uncalcined rhodochrosite and Philipsburg lump (lot 2b). On account of the high loss (31 per cent) on ignition of the rhodochrosite, it gave trouble, even when mixed with the other ore. The large volume of CO_2 given off by the raw rhodochrosite tended toward excessive electrode consumption, and probably also toward wasting of the reducer in the charge. If ore so high in volatile matter falls into the melt before it has been thoroughly calcined in the shaft, the furnace will "blow," or form a "volcano."

For these reasons, and because it should be cheaper to calcine by fuel heat than to expel CO_2 by electric heat in the furnace, it will probably be found desirable to calcine rhodochrosite ore before charging it into the electric furnace. The ore decrepitates somewhat on calcination, and the product may have to be screened, the coarse material being then smelted continuously and the fines fed to a separate furnace as fast as the melt will take the ore, but without keeping the shaft piled full, in order to keep down dust losses. Rhodochrosite, may, in fact, be too high grade an ore for electric smelting without being mixed with ores containing more slag-forming constituents.

The analysis of the rhodochosite used, after complete calcination, was 53.5 per cent Mn, 2.9 per cent Fe, 8.8 per cent SiO_2, 1.65 per cent

$CaO + MgO$, 1.4 per cent Al_2O_3, and 0.04 per cent P. Fluxed to the ordinary basic slag and with suitable addition of steel scrap, this would give only about 1,050 pounds of slag per long ton of 80 per cent ferromanganese produced. Compared to the 40 per cent Mn 20 per cent SiO_2 ore in common use, which, when similarly slagged, gives about 4,000 pounds of slag per long ton of alloy, the rhodochrosite may cause too short a column of slag, and hence, at the low voltage of a ferromanganese furnace, may not present sufficient electrical resistance to develop enough heat.

In the ordinary three-phase, nonconducting-bottom furnace, the current may flow from one electrode to the other two by two paths—(1) through slag entirely, or (2) from the electrode straight down through the slag to the alloy beneath and then through the alloy to directly beneath the other electrodes, and up through the slag again to the other electrodes. With too short a column of slag, path 1 presents the least resistance and will result in the generation of heat in the slag in three spots only. With a deeper slag column, the resistance of the path from one electrode to another through the slag only is less than that from electrode to electrode by way of slag, alloy, and slag. If the power input is too low, the large mass of metal below the slag may get cool enough to make tapping difficult, and the slag itself may not get hot enough for complete reduction of manganese, resulting in high manganese loss in the slag. If, in order to develop enough heat, the furnace is run as an arc furnace instead of as a resistance furnace, local overheating will occur and manganese will be lost by volatilization. In this connection, the comments of Bardwell[a] on smelting rhodochrosite are of interest.

TESTS OF EFFECT OF SIZE OF REDUCER CONSTITUENTS.

Runs 50 and 52, previously mentioned, had shown that the mixture of half coke with half charcoal worked as well as all charcoal. When all one-fourth-inch coke was used as reducer, some of the excess of fine coke tended to emulsify with the slag, coming out with it and making it harder to tap. When half coke and half charcoal was used this was not the case. It was thought that larger coke might give better results, so a series of runs was made to test this assumption and to compare the results from coke and from anthracite with the results of runs in which a mixture of half coke with half charcoal was used. The results of the tests are presented in the table following. The ore used was Philipsburg screenings, lot 4, containing 26.4 per cent Mn, 2.5 per cent Fe, 36.4 per cent SiO_2, 7.5 per cent $CaO + MgO$, 1.9 per cent Al_2O_3, and 0.21 per cent P, with a 11.9 per cent loss on ignition. It was crushed to pass a one-half-inch screen. As a flux, only 14.5 pounds of CaF was used per 100 pounds of ore. Three heats were made on each run; that is, 300 pounds of ore was used in each run.

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[a] Bardwell, E. S., Discussion; Bull. 43, Am. Inst. Mining Eng., Nov., 1918, p. 1651.

Results of tests of reducers to determine effect of size of constituents.

Run No.	Coke [a] (½-inch).	Charcoal (½-inch).	Coke [b] (1 to ½ inch).	Coke [b] (less than ½-inch).	Anthracite [c] (¾ to ½ inch).	Anthracite [d] (less than ½-inch).	Recovery of Mn.	Recovery of P.
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*Per cent.*	*Per cent.*
68	10	10	....	....	....	....	71	75
71	....	....	21	....	....	....	54	68
74	....	....	....	21	....	....	35	....
72	....	....	....	....	21	....	46	65
75	....	....	....	....	....	21	32	53

Run No.	Time.	Power used, kw.-hours.	Alloy made.	Composition of alloy.			Slag made.	Mn in slag.	P.	Dross.		
				Mn.	Si.	P.				Weight.	Mn content.	P content.
	Hr. min.		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*
68	7 20	452	78.8	71.0	15.5	0.60	177	12.0	0.04	5	(e)	(e)
71	7 15	450	60.8	71.0	16.5	.71	156	15.4	.06	21	14.9	0.13
74	6 45	450	57.6	73.5	13.2	(f)	183	15.0	(f)	29	21.0	(f)
72	7 0	450	50.7	72.0	15.0	.81	200	15.6	.03	29	13.9	.31
75	6 55	450	[g] 33.6	73.6	11.5	1.00	205	13.1	.03	66	20.7	.14

[a] Metallurgical coke, 4.75 per cent H_2O, 10.3 per cent ash.

[b] Gas coke, 13.8 per cent ash.

[c] 4.5 per cent H_2O, 8.5 per cent ash.

[d] 3.5 per cent H_2O, 13 per cent ash.

[e] Included in slag.

[f] Not determined.

[g] The fine coal lying on the slag at the end of a tap ran out with the slag, leaving less excess reducer in the furnace on the last two heats than in the other runs. In run 75, 20 to 24 pounds of dross was obtained on each tap. In the other runs the dross was all taken out after the last tap only.

These results indicate that anthracite is not as good a reducer as coke, perhaps because anthracite is less porous, giving less area of contact with the melt. It would be interesting to know whether the difference in the performance of coke and anthracite would be as marked in a large furnace as it was in the experimental one. The experimental work indicates that, at least with a highly siliceous ore, the mixture of coke and charcoal used by the Noble Electric Steel Co. should be highly desirable where charcoal is available.

SLAG-SMELTING TESTS.

In the summer of 1917 Prof. F. F. McIntosh, consulting chemist of the Bureau of Mines, and assistants, made some tests at the Carnegie Institute of Technology, Pittsburgh, Pa., on making manganese alloys from slags. A 50-kw. furnace was used. The slags were not fluxed, and as the iron and manganese were reduced out the slags became stiff, so that the furnace had to be torn down after each tap. The results, therefore, could hardly be applied directly to commercial practice. Table 30 gives a summary of some of the results.

TABLE 30.—*Results of slag smelting tests in electric furnace.*

Run No.	Slag or slag mixture.	Kind of slag.	Analysis of slag or mixture.					Reducer.		Time.	Kw.-hours used.	Alloy made.	Composition of alloy made.				Recovery of Mn.
			Mn.	Fe.	SiO_2.	CaO.	Al_2O_3.	Coke.	Charcoal.				Mn.	Fe.	Si.	P.	
	Lbs.		*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*Lbs.*	*Hr. min.*		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
G	240	Bessemer ladle	8.5	13.5	57	2.5	7	44	....	7 45	260	59.5	23	50	26	....	65
H	1,090	Bessemer ladle plus basic open-hearth.	11	21	42	12	10	178	....	22 0	900	108.5	14.5	71	12	0.50	13
J	536	Bessemer ladle plus manganese steel plus spiegel cupola slags.	24	7	42	6	11	20	116	14 45	650	126.5	52.5	26	19	.11	53
K	548	do	22	5.5	45	6	12	....	126	13 15	660	142.0	53	23	21	.14	63
L	280	Manganese steel plus spiegel cupola slag.	26	6.5	40	[a] 3	12	122	....	6 15	320	51	51.5	27	17	.06	36

[a] 19 pounds of CaO also added to the charge.

Runs K and L indicated that charcoal was a better reducer than coke. Evidently such manganese-bearing slags need to be fluxed, so as to make a fluid slag, as they contain 11 to 22 per cent $Al_2O_3 + CaO$, but those of about 25 per cent Mn content could be used to make silicomanganese. The basic open-hearth slag used in run H was rather high in phosphorus. The other slags were low in phosphorus, and hence made alloys very low in that element.

EFFECT OF PHOSPHORUS CONTENT OF ORES.

In the experimental runs low-grade ores high in phosphorus gave an alloy rather high in that constituent. If smelted with the shaft kept full of ore, the alloy might be still higher in phosphorus. A considerable loss of phosphorus by volatilization took place, and the slags were always rather low in phosphorus, no matter what the original content in the ore was, or whether the slag was acid or basic. The percentage recoveries of phosphorus and of manganese in the alloy were usually within a few per cent of each other.

Lyon, Keeney, and Cullen[a] state that in the electric smelting of ferromanganese all the phosphorus in the charge goes into the metal, and that in making ferrosilicon most of it does, owing to the strongly reducing conditions.

Newton[b] makes a similar statement in regard to blast-furnace ferromanganese smelting. Continuous smelting with the shaft kept full of ore will tend to condense any volatilized phosphorus, so that the losses found in the experimental smelting by batches may not occur in normal practice. Doubtless the lower the ore is in phosphorus the less will escape. The remarks cited are, of course, based on normal practice with low phosphorus ores. However, W. W. Clark, former manager of the Noble Electric Steel Co., states[c] that phosphorus is volatilized in the Noble ferromanganese furnace.

Lonergan[d] cites a test run at the Iron Mountain Alloy Co. works on ore having a very low phosphorus content (less than 0.04 per cent P) in which 52.6 per cent of the phosphorus contained in the ore, coal, and limestone was recovered in the ferromanganese, 14.75 per cent in the slag, and 6.06 per cent in "dirty metal and slag," leaving 31.4 per cent phosphorus lost by volatilization and in dust. Lonergan concluded that the greater part of this loss is purely mechanical, the phosphorus in the dust being retained in its original form. He assumes that any volatilized phosphorus would go off as phosphine.

The writers see nothing to prevent elemental phosphorus being volatilized, as it is in electric-furnace production of phosphorus, and burning to the oxide as soon as it reaches the air.

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[a] Lyon, D. A.; Keeney, R. M.; and Cullen, J. F.: The electric furnace in metullargical work: Bull. 77, Bureau of Mines, 1917, pp. 144, 166.

[b] Newton, E., Manganiferous iron ores of the Arizona District, Minn.: Bull. 5, Minn. School of Mines Exp. Station, Univ. of Minn., 1918, p. 75.

[c] Personal communication.

[d] Lonergan, J., Eliminating phosphorus and sulphur in electric ferromanganese furnaces: Met. and Chem. Eng., vol. 20, 1919, p. 245.

If 31.4 per cent of the manganese also was lost in dust, to say nothing of volatilization, then Lonergan's assumption that phosphorus was lost only mechanically would be justified; but according to Keeney,[a] the average loss by volatilization and dusting in the Iron Mountain Alloy Co.'s practice is 7.3 per cent. It would therefore appear that even on ores so low in phosphorus an appreciable amount of the phosphorus is volatilized.

TESTS WITH HIGH-PHOSPHORUS ORES.

Runs 60 and 61 were made with Philipsburg lump ore, lot 3b, containing 24.1 per cent Mn, 1.6 per cent Fe, 37.7 per cent SiO_2, 7.7 per cent CaO+MgO, 1.6 per cent Al_2O_3, and 0.37 per cent P, with 12.8 per cent loss on ignition. The procedure and results were as follows:

Results of run 60—discontinuous smelting, each charge being tapped before adding the next.

[Furnace cold at start.]

	Charge a.	Charge b.	Total for run.
Make-up of charge:			
Ore......pounds..	100	100	......
$CaCO_3$ (0.32 pound CaO+O per pound of SiO_2)......do....	8	8	......
CaF_2 (18 pounds per pound of SiO_2)......do....	7	7	......
Coke......do....	7.5	8.5	......
Charcoal......do....	7.5	8.5	......
Steel scrap......do....	0.5	None.	......
Time of run......	3 h. 25 m.	2 h. 25 m.	5 h. 50 m.
Power used......kw.-hours..	149	130	279
Metal tapped......pounds..	6.0	18.7	24.7
Slag tapped[a]......do....	65.0	70.5	135.5

Analysis of metal:	*Per cent.*	Distribution of metal:	*Per cent.*
Mn......	66.0	Mn......	34.0
Si......	13.5	P......	27.5
P......			
Analysis of slag:	0.71	Distribution of slag:	
Mn......	15.0	Mn......	42.5
P......	0.7	P......	14.5

[a] Some slag stuck in furnace.

Results of run 61—continuous smelting, shaft being kept full.

[Total charge in pounds: Ore, 400; $CaCO_3$ (0.35 pound CaO per pound of SiO_2), 40; CaF_2 (1.18 pound per pound of SiO_2) 28; coke, 38.5; charcoal, 38.5.]

Tap No.	After.	Kw.-hours used.	Metal tapped.	Analysis of metal.			Slag tapped.	Analysis of slag.	
				Mn.	Si.	P.		Mn.	P.
	Hr. min.		*Pounds.*	*Per ct.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*
1......	2 30	142	13.5	66.5	11.6	0.80	92.8	17.1	0.07
2......	1 50	141	30.1	70.5	16.5	.95	84.0	13.2	.09
3......	2 55	141	31.1	70.0	18.3	.93	30.0	11.8	.12
4......	40	28	.8	......	......	......	[a]42.5	10.5	.15
Total......	7 55	452	75.8	......	......	......	......	......	......
Average......	......	......	......	69.0	16.0	.93	249.3	14.0	.095

Distribution.	Mn.	P.
	Per cent.	*Per cent.*
Metal......	54.5	57.5
Slag......	35.5	21.5
Loss......	10.5	21

[a] Includes dross.

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[a] Keeney, R. M., The manufacture of ferroalloys: Bull. Am. Inst. Min. Eng., August, 1918, p. 1333.

These results, compared with those from run 68, show that the power supply was insufficient for good recovery of the manganese, 452 kw.-hours being used on 400 pounds of ore in run 61 and 452 on 300 pounds of ore in run 68. The reducer was probably not in too small excess, and a slag higher in CaF_2 would have given better results. Under proper conditions this ore should give the 71 per cent recovery of run 68, when the increased reduction of manganese and silicon should bring the phosphorus down to about 0.75 per cent.

TESTS ON VOLATILIZATION OF PHOSPHORUS.

As the volatilization of phosphorus is probably due to to its reduction from the calcium phosphate in the slag to elemental phosphorus, which will then either volatilize or be absorbed by the metal, it would appear that if only enough reducer were used to reduce Fe and Mn to FeO and MnO, and P_2O_5 to P, and the charge were held molten for a time before more reducer were added to cause a metal fall, the elimination of phosphorus would be favored. With this consideration in mind, run 62 was made.

In this run each of the three batches charged consisted of 100 pounds of lot 2b ore, 10 pounds of $CaCO_3$, 7 pounds of CaF_2, and 2 pounds each of coke and charcoal. After the batch was melted it was kept molten for 20 minutes, then 8 pounds each of charcoal and coke were added. The results were as follows:

Results of run 62.

Tap.	Time.		Kw.-hours used.	Quantity of metal obtained.	Analysis of metal.			Quantity of slag obtained.	Analysis of slag.		Quantity of dross obtained.	Analysis of dross.	
					Mn.	Si.	P.		Mn.	P.		Mn.	P.
	Hr.	*min.*		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*
a	3	5	182	13.5	......	......	1.29	67	13.3	0.06	......	......	......
b	2	35	131	19.8	......	......	.97	78	11.8	.06	......	......	......
c	2	20	130	15.0	......	......	1.16	70	11.7	.10	27	17.1	0.16
Total	8	0	443	48.3	......	......	......	215	......	......	......	......	......
Average	......		......	......	71.0	14.0	1.12	......	12.6	.07	......	......	......

Distribution:

		Mn.	*P.*
Metal	per cent	42	49
Slag and dross	do	43	31
Loss	do	15	20

As there was a considerable excess of reducer left after each tap, which would cause too early a metal fall and collect phosphorus, another run (No. 63) was made, omitting the reducer from the charge till all had been melted 20 minutes (except on the first charge), letting the excess reducer remain in the furnace after each tap, and then adding 10 pounds of coke and 10 pounds of charcoal. Five pounds of NaCl was added to each charge. Otherwise the charges were as in run 62. The results obtained were as follows:

Results of run 63.

Tap.	Time.	Kw.-hours used.	Quantity of metal obtained.	Analysis of metal.			Quantity of slag obtained.	Analysis of slag.	
				Mn.	Si.	P.		Mn.	P.
	Hr. min.	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*
a	3 15	182	13.0	......	......	1.42	72	12.0	0.02
b	3 30	136	20.6	......	......	.75	81	11.9	.03
c	2 15	131	21.2	......	......	.80	93	11.0	[a].03
Total	9 0	449	54.8	......	......	......	246	......	......
Average	......	......	......	71.6	14.5	.93	......	11.5	.03

[a] Slag included in dross.

Distribution:

		Mn.	P.
Metal	per cent	54.5	46
Slag	do	39.5	6
Loss	do	6	48

On the assumption that a more acid slag might help, another run (No. 64) was made, the total charge being practically the same as in runs 62 and 63, but divided as follows:

Charge for run 64, in pounds.

	Batch a.	Batch b.
Added at start:		
Ore	100	100
Charcoal	4	None.
$CaCO_3$	3	5
CaF_2	2	1
Added after heating 20 minutes:		
Charcoal	6	10
Coke	10	10
$CaCO_3$	7	8.5
CaF_2	5	6

Results of run 64.

Tap.	Time.	Kw.-hours used.	Quantity of metal obtained.	Analysis of metal.			Quantity of slag obtained.	Analysis of slag.		Dross.
				Mn.	Si.	P.		Mn.	P.	
	Hr. min.		*Pounds.*	*Per ct.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*	*Lbs.*
a	4 30	210	9.3	68.0	12.8	1.45	65	18.0	0.11	......
b	2 35	148	10.9	71.8	11.8	1.57	63	18.0	.09	17
Total	7 5	358	20.2	......	......	......	128	......	......	......
Average	......	......	......	70.0	12.5	1.52	......	18.0	.10	......

Distribution:

		Mn.	P.
Metal	per cent	29.5	42
Slag	do	48	17.5
Loss	do	22	40.5

As the results of this run were not promising, another run (No. 59) was made in which the excess reducer was carefully scraped out after each tap, so that only the four pounds of reducer charged with the ore, as in run 62, would be present.

In run 59, the charge at the start consisted of 100 pounds of ore (lot 2b), 8 pounds of $CaCO_3$, 5 pounds of CaF_2, and 4 pounds of charcoal. After all was melted for 20 minutes, 6 pounds of charcoal and 6 pounds of coke were added; then the furnace was tapped. This procedure was repeated four times. The results follow:

Results of run 59.

Tap.	Time.	Kw.-hours used.	Quantity of metal recovered.	Analysis of metal.			Quantity of slag and dross recovered.	Analysis of slag and dross.	
				Mn.	Si.	P.		Mn.	P.
	Hr. min.		*Pounds.*	*Per ct.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*
a	4 0	200	12.5	66.0	11.9	0.70	83.3	17.0	0.10
b	2 50	130	10.2				80.5		
c	2 45	116	9.6	66.2	12.3	.64	85.7	17.1	.06
Total	9 35	446					249.50		
Average			32.3	66.0	12.0	.68		17.0	.09

Distribution:

		Mn.	P.
Metal	per cent	29.5	19.5
Slag	do	58.5	20
Loss	do	12	60.5

These results, owing to the poor slag and the lack of sufficient excess reducer, are far from satisfactory as regards recovery of manganese, but they are somewhat promising as regards volatilization of phosphorus.

The low recoveries of manganese in this series were doubtless largely due to the ore not being properly fluxed. This defect was not realized till the later runs on fluxing were made. However, it is evident that one requirement for high recovery of manganese, namely, an excess of reducer, and one for high volatilization of phosphorus, namely, no excess reducer beyond that necessary to form MnO, FeO and P (that is, no metal fall to collect phosphorus), are incompatible. It is hardly practicable to clean the furnace from all excess carbon after each tap.

DEPHOSPHORIZATION AND SMELTING IN TWO STAGES.

Therefore, the only procedure metallurgically possible seems to be to carry out the dephosphorization in one furnace, running with just enough reducer to form MnO, FeO, and P, and holding the charge molten for a while to allow the phosphorus to escape; then to tap the hot slag into a second furnace, add excess reducer, and smelt the previously dephosphorized slag.

As it would be difficult to determine the exact point where there is complete reduction to MnO, FeO, and P, and as the electrode will also supply a little carbon, it would probably be more feasible to provide the dephosphorizing furnace with metal and slag taps, the latter being at a higher level. The slag can thereby be tapped into the second furnace, and a very little high-phosphorus metal can be taken either at each slag tapping or at longer intervals. This scheme would eliminate phosphorus not only by volatilization but also, in part, by concentration into the first metal to fall.

That this method is metallurgically possible is shown by the results of runs 56 and 57, presented in the following tabulations:

Charges used in run 56.

[Figures in pounds.]

Charge.	Ore.[a]	Charcoal.	Coke.	$CaCO_3$.	CaF_2.	Steel scrap.	Mill scale.
a	100	9.5	9.5	15	6		
b	100	2.5	2.5	8	5	10	
c	100	2.5	2.5	8	5		
d	100	5.5	5.5	6	5		15
e	75	4	4	5	4		11

[a] From lot 2b.

NOTES.—
a. Run to reduce considerable Mn.
b. Run to reduce very little Mn, steel scrap added to collect P.
c. Run to reduce very little Mn.
d. Run to reduce very little Mn, and leave Fe in slag from mill scale.
e. Run to reduce very little Mn, but all the mill scale.

Charges used in run 57.

Charge.	Ore.	Charcoal.	$CaCO_3$.	NaCl.
a	50	1.9	6.6	4.3
b	50	1.9	6.6	4.3

NOTES.—
a. Run to reduce very little Mn.
b. Run to reduce a good deal of Mn.
It was hoped that the addition of NaCl would form phosphorus chlorides or oxychlorides that might be more readily volatile.

Results of runs 56 and 57.

Run.	Time.		Kw.-hours used.	Quantity of alloy made.	Analysis of alloy.		Slag.	Analysis of slag.	
					Mn.	P.		Mn.	P.
	Hr.	*min.*		*Pounds.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*
56a	3	15	136	13.2	66.5	0.96	74.8	15.8	0.05
56b	1	25	73	12.8	41.1	1.02	99.5	22.0	.12
56c	1	40	60	2.0	69.6	1.41	87.0	23.0	.11
56d	1	20	59	2.6	18.8	1.23	86.0	21.6	.10
56e	2	15[a]	105	11.1	41.8	1.32	59.0	21.6	.05
57a	1	10[b]	41	.4	32.3	1.07	49.0	24.0	.09
57b	2	35	88	9.0	50.1	.39	41.0	16.6	.11

[a] Furnace cold at start. [b] Furnace hot at start.

Results of runs 56 and 57—Continued.

DISTRIBUTION OF MN. AND P.

Run.	Mn.			P.		
	Alloy.	Slag.	Loss.	Alloy.	Slag.	Loss.
	Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
56a	37	49	14	34.5	11	53.5
56b	17	91	[a]8	34.5	32.5	33
56c	6	83	11	7.5	26 [b]41	52.5
56d	2	78	20	9	23 [b]38	53
56e	26	74	0	40	7.5	52.5
57a	1	97.5	1.5	2	24.5	72.5
57b	33	56	11	19.5	24.5	56

[a] Gain. [b] Including dross.

The results of runs 56b, 56c, and 57a indicate that by proper care 85 to 90 per cent of the manganese could be recovered in the slag, with only about 25 to 30 per cent of the phosphorus, the slag having about 23 per cent manganese and 0.10 per cent phosphorus where the original ore had 24 and 0.37 per cent, respectively, or, on the basis of the calcined ore (deducting loss on ignition), 27.5 and 0.42 per cent.

Attempts to smelt the dephosphorized slags without the proper flux and with too little reducer gave low recoveries of manganese. After tests had indicated the flux and reducer needed, similar tests were again made. Not enough ore from lot 2b was left for a run, so ore from lot 4 was added. The charges were as follows:

In run 76 (dephosphorization), 57 pounds of lot 4 ore, averaging 25.4 per cent Mn, 2.1 per cent Fe, 37.2 per cent SiO_2, and 7.6 per cent $CaO+MgO$; 43 pounds of lot 2b ore, averaging 1.7 per cent Al_2O_3, 0.29 per cent P, and 12.1 per cent loss on ignition; 4 pounds of charcoal; and 7½ pounds of CaF_2, were charged in two batches. The results were as follows:

Results of run 76 (dephosphorization).

	Charge 1.	Charge 2.
Time	2 h. 10 m.	1 h. 35 m.
Kw.-hours used	125	100
Alloy made, pounds	2.9	4.5
Manganese in alloy, per cent	54.	64.1
Phosphorus in alloy, do	2.5	2.12
Slag made, pounds	96.7	82
Manganese in slag, per cent	23.4	20.3
Phosphorus in slag, do	.1	.09
Distribution of manganese:		
Metal, per cent	9	
Slag, do	79	
Loss, do	12	
Distribution of phosphorus:		
Metal, do	29.5	
Slag, do	29.5	
Loss, do	41	

The second tap was run a little too long and too much metal reduced.

The slags were then smelted. In order to determine what the results would be if both slags and metal were run into the smelting furnace from the dephosphorizing furnace, the metals as well as the slags were recharged.

In run 77 (smelting) the first charge consisted of 96.7 pounds of slag, 2.9 pounds of metal from run 76 (1), 17 pounds of charcoal (charcoal only was used, the coke supply being exhausted), and 7½ pounds CaF_2. The second charge consisted of 82 pounds of slag, 4.5 pounds of metal from run 76 (2), 17 pounds of charcoal, and 7½ pounds CaF_2. The results were as follows:

Results of run 77 (smelting).

	Charge 1.	Charge 2.	Total (or average.)
Time	2 h. 35 m.	1 h. 55m.	4 h. 30 m.
Kw.-hours used	175	130	305
Metal recovered ....pounds..	23.7	21.5	45.2
Manganese in metal ....per cent..	75.2	72.5	74
Silicon in metal ....do	....	....	17.2
Phosphorus in metal ....do....	.53	.53	.53
Slag recovered ....pounds..	50	66	116
Manganese in slag ....per cent..	9.8	8.8	9.2
Phosphorus in slag ....do....	.06	.07	.65

Based on the manganese and phosphorus in the slag and metal charged in run 77, the distribution was as follows:

Percentage distribution of Mn and P based on analysis of slag and metal charged.

	Mn.	P.
Metal	75	70.5
Slag	24	20.5
Loss	1	9

Based on the analysis of original ore charged in run 76, the distribution when both slag and metal of that run were recharged was as follows:

Percentage distribution of manganese and phosphorus, based on original ore.

	Mn.	P.
Final metal	66	41.5
Final slag	21	12
Loss, dephosphorizing run	12	41
Loss, smelting run	1	5.5

The fact that the manganese loss was 12 per cent in the first (dephosphorizing) run, as compared with 1 per cent in the second (smelting) run, although the temperature was much higher in the second, indicates that the loss in smelting is more by dusting than by volatilization. The true volatilization loss appears to be small as long as the furnace is run as a resistance furnace, or with only a small submerged arc. A large, open, high-voltage arc would doubtless volatilize manganese.

To find what the results would have been had the metal obtained in run 76 not been recharged in run 77, we may subtract the 7.4 pounds of alloy, with its content of 4.4 pounds of manganese and 0.17 pound of phosphorus, from the 45.2 pounds of alloy obtained in run 77, with its 33.4 pounds of manganese and 0.24 pound of phosphorus. This would leave 37.8 pounds of alloy, containing 29 pounds of manganese, 7.75 pounds of silicon, and 0.07 pound of phosphorus, and analyzing about 77 per cent of manganese, 20 per cent silicon, and 0.18 per cent phosphorus, most of the iron being taken out in the first alloy.

The distribution, based on the content in the original ore, would then have been:

Percentage distribution of Mn and P when the alloy is not resmelted.

	Mn.	P.
Final metal	57	12
Final slag	21	12
Loss, dephosphorizing run	12	41
Loss, smelting run	1	5.5
Discarded in first metal	9	29.5

If the second charge in heat 76 had not been run so long, so that only 2 or 3 pounds of metal instead of 4½ pounds had been collected, the recovery, based on the original ore, would have been about 60 per cent.

However, if the high phosphorus alloy charged in run 77 had been left out, more of the phosphorus left in the dephosphorized slag might have gone into the metal, as the phosphorus may be expected to divide itself between the two liquid layers somewhat according to the concentration of phosphorus in each, the recovery of the 0.075 pound of phosphorus left in the two slags might have been practically complete. This would have given, in 37.8 pounds of alloy, 0.20 per cent phosphorus.

Two more slags were made from the same charges used in run 76, but the reduction was carried further, more metal being reduced than in run 76. The slags contained an average of 20 per cent manganese and 0.10 per cent phosphorus. These were smelted without the addition of the metal obtained in the dephosphorizing run and gave an alloy of about 72 per cent manganese, 27 per cent silicon, and 0.17 per cent phosphorus, with a recovery of 71 per cent manganese, based on the manganese content in the slags charged.

As in runs 56 and 57, the slags from an ore containing 0.37 per cent phosphorus were brought down to 0.10 per cent phosphorus in the dephosphorizing run, and as in run 76 an ore containing 0.29 per cent phosphorus came down also to 0.10 per cent phosphorus, it is probable that the slag of ores containing even more than 0.37 per cent phosphorus could also be brought down to about the same phosphorus content without much greater loss of manganese.

In order to get an idea of the power that would be needed in smelting the hot dephosphorized slag, the power consumption was noted in run 77 from the time the slag was thoroughly melted—that is, in about the condition it would be when tapped from the dephosphorizing furnace till the end of the heat. On the first heat, starting with a cold furnace, 95 kw.-hours was used and on the second 85 kw.-hours was used. On a third heat 70 kw.-hours would probably suffice.

The dephosphorization heats in run 76 were doubtless continued longer than necessary, as in heats c and d, run 56, and only 60 kw.-hours was needed.

CONCLUSIONS AS TO TWO-STAGE PROCESS.

It would appear that with a 3,000-kw. installation, using a 1,350-kw. furnace for dephosphorization and a 1,650-kw. furnace for smelting, 12,500 pounds of ore (containing 24 per cent Mn, 1.5 per cent Fe, 38 per cent SiO_2, 7.5 per cent CaO + MgO, 0.4 per cent P, and 1.5 per cent Al_2O_3, with 13 per cent loss on ignition), mixed with 250 pounds of coke, 250 pounds of charcoal, and 940 pounds of fluorspar (CaF_2), and given 5,500 kw.-hours, would produce 250 pounds of high phosphorus alloy, with about 50 per cent Mn and 2.5 per cent P, containing about half of the Fe in the ore and about 11,000 pounds of slag, with 23 per cent Mn and 0.10 per cent P. The latter tapped hot into the second furnace and mixed with 950 pounds of coke, 950 pounds of charcoal, and 860 pounds of fluorspar, and smelted with 7,000 kw.-hours, should produce 2,350 pounds (equivalent to a long ton of 80 per cent alloy) of an alloy containing 76 per cent Mn, 4.5 per cent Fe, 18.3 per cent Si, 1.0 per cent C, 0.20 per cent P, with a recovery of 60 per cent of the metallic manganese in the original ore.

The ore smelted direct without dephosphorization would give an alloy containing about 73 per cent Mn, 8 per cent Fe, 17 per cent Si, 1.25 per cent C, and 0.75 per cent P, with a recovery of 70 per cent of the manganese. A 3,000 k. v. a. installation including two furnaces as suggested above would produce daily 4.5 long tons of the dephosphorized alloy, containing 7,700 pounds of Mn, 1,850 pounds of Si, 456 pounds of Fe, 100 pounds of C, and 20 pounds of P, while a single 3,000 k. v. a. furnace would produce daily 5.5 long tons of the high-phosphorus alloy, containing 9,000 pounds of Mn, 2,100 pounds of Si, 950 pounds of Fe, 185 pounds of C, and 92 pounds of P.

Whether the lower phosphorus content resulting from the two furnace treatments would compensate for the lower manganese recovery and the lower output depends on whether the high-phosphorus alloy can be utilized.

If 0.9 per cent of standard 80 per cent ferromanganese—that is to say, 0.72 per cent ferromanganese—is added to the average steel bath, a content of 0.4 per cent phosphorus in the ferromanganese (the ratio of P to Mn being 1 to 200) would raise the phosphorus content of the steel by 0.0036 per cent. Ferro alloy of this ratio has been accepted by some steel makers during the manganese shortage.

The alloy containing 73 per cent manganese, 17 per cent silicon, and 0.75 per cent phosphorus would raise the phosphorus content of the steel by 0.0074 per cent, while the alloy containing 76 per cent manganese, 18.3 per cent silicon, and 0.2 per cent phosphorus would raise it only 0.0019 per cent. Either alloy would raise the silicon content of the steel 17 points, or a little less, as some silicon will be lost by oxidation.

An indefinite quantity of manganese alloys high in phosphorus can doubtless be absorbed by the steel industry. Cromlish,[a] in fact, suggested making a spiegel with 13 to 25 per cent manganese and 2.2 to 3.4 per cent phosphorus from "flush" and "tapping" cinders and utilizing this in the manufacture of the steel sheet, where a phosphorus content higher than normal is required to keep the sheets from sticking together in pack-rolling.

During the preliminary runs, and in some runs not reported herein, with ores high in phosphorus, when the proper flux or reducer was not used, a number of alloys running from 65 to 75 per cent Mn, 3 to 17 per cent Si, 1.3 to 5.3 per cent C, and 1.6 to 0.7 per cent P were produced. An assortment of these alloys has been sent to the Minneapolis station of the Bureau of Mines, where it is planned to study the possibility of eliminating the phosphorus.

According to Lang,[b] by melting ferromanganese high in phosphorus under manganese oxide (MnO_2) at 1,200° C., the phosphorus may be eliminated. A couple of tests were made at Ithaca by melting the ferro alloys high in phosphorus under high-grade manganese-oxide ores. No dephosphorization of the alloy, but instead a dephosphorization of the ore occurred.

TESTS WITH HIGH-SILICA ORES.

An important class of low-grade manganese ores is that in which manganese and iron occur in about equal amounts, with or without a high proportion of silica or phosphorus, or both, to manganese.

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[a] Cromlish, A. L., U. S. Patent 1261907, Apr. 9, 1918.

[b] Lang, G., German Patent 252166, class 18 b, group 2, Oct. 14, 1912.

Many Cuyuna ores, such as the products of the Ferro and Mille Lacs mines, some of the Appalachian ores, and some ores from Leadville, Colo., fall into this class.[a]

A small lot of such an ore was at hand from the Sultana mine, Cuyuna district. The lump ore analyzed 29.0 per cent Mn, 24.2 per cent Fe, 4.5 per cent SiO_2, 1.5 per cent CaO +MgO, 3.7 per cent Al_2O_3, 0.20 per cent P, and showed 11.6 per cent loss on ignition.

Such an ore, smelted direct, would give about a 50 per cent manganese alloy. The ratio of phosphorus to manganese is high. It seems desirable to produce manganese alloys either of the composition of spiegel, which will bear cupola melting, for adding hot without excessive loss of manganese; or of that of 70 to 80 per cent ferromanganese for adding cold. Alloys of intermediate composition, say 40 to 60 per cent manganese, where the balance is iron instead of silicon, will chill the steel bath badly if added cold on account of the large amounts of inert iron carried. In order to add such alloys hot, electric melting of the alloys is necessary, as they would lose too much manganese in cupola melting. Electric melting, even of 80 per cent ferromanganese, is probably desirable, but few American steel plants are doing so as yet, and the necessity of installing electric furnaces in order to utilize the 50 per cent manganese alloys would be an obstacle to the adaption of electric melting. Where silicomanganese containing 60 per cent manganese and 20 per cent silicon is used instead of both ferromanganese and ferrosilicon, the inert iron carried by the silicomanganese is less than that in an equivalent mixture of 80 per cent ferromanganese and 50 per cent ferrosilicon.

APPLICABILITY OF TWO-STAGE PROCESS TO CUYUNA ORES.

As so many Cuyuna ores carry an undesirably high proportion of phosphorus to manganese, it appears that if electric smelting is used, a differential, two-stage, two-furnace process is worth consideration. The bulk of the iron, a little manganese, and much of the phosphorus may be reduced in the first furnace to a high-phosphorus, manganiferous pig iron, or, in an ore sufficiently low in phosphorus, to a spiegel, leaving in both cases a slag high in manganese, low in iron, and low in phosphorus to be tapped hot into the second furnace and there smelted to ferromanganese.

RESULTS OF TESTS.

Tests with the two-stage process were made with Cuyuna ores. In run 54, three heats were made, each charge including 100 pounds of Sultana lump ore, 3¾ pounds limestone (the furnace bottom had been patched with dolomite before this run, and some of the dolomite went into the slags), and 12 pounds of charcoal (except in heat c, where 10 pounds was used). The results were as follows:

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[a] Harder, E. C., Manganiferous iron ores: Bull. 690, U. S. Geol. Survey, 1918, pp. 5, 8, 11.

Results of run 54.

Heat.	Time.		Kw.-hours used.	Alloy.	Analysis of alloy.					Slag.	Analysis of slag.		
					Mn.	Fe.	Si.	C.	P.		Mn.	Fe.	P.
	H.	*min.*		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*
a	2	15	100	30.2	24.5	68.1	0.07	2.35	0.17	60.0	32.0	2.7	0.04
b	1	20	58	20.3	11.4	82.5	.23	1.7	.19	41.0	44.8	5.9	.06
c	1	20	50	20.0	10.5	84.0	.02	2.25	.19	51.0	46.9	4.4	.03
d[a]	0	15	10	13.0	17.1	75.3	.40	2.10	.21	21.0	36.1	3.0	Tr.
										[b]15.5	26.1	13.8	.08

DISTRIBUTION OF Mn Fe, AND P.

	Mn.	Fe.	P.
	Per cent.	*Per cent.*	*Per cent.*
In metals	13.5	75	22
In slags	75.5	12	13
Volatilization and dust loss	11.0	13	65

[a] A little CaF_2 was added and a longer run was made to get the slightly stiff slag to tap more cleanly from the furnace.

[b] Mixed slag and metal shot taken from furnace bottom when cold.

The reduction in heat a was carried too far, reducing too much Mn. With experience it should be possible to leave about 80 per cent of the Mn, 15 per cent of the Fe, and 20 per cent of the P in the slag; take out 10 per cent of the Mn, 75 per cent of the Fe, and 20 per cent of the P in the metal; and lose 10 per cent Mn, 10 per cent Fe, 60 per cent P by volatilization and dusting.

Two thousand pounds of ore plus 200 pounds of reducer (half charcoal, half coke) plus 200 pounds of limestone, smelted with 1,000 kw.-hours or less, would then produce about 425 pounds of alloy (low-grade spiegel) containing 13 per cent Mn, 89.5 per cent Fe, 2 per cent C, and 0. 20 per cent P; and 1,100 pounds of slag containing 42 per cent Mn, 6.5 per cent Fe, 8 per cent SiO_2, and 0.07 per cent P, that is the equivalent of a high-grade ore.

This slag tapped hot into the second furnace, with the addition of 50 pounds of limestone and 330 pounds of reducer (half coke, half charcoal), and smelted with 700 kw.-hours, should give, assuming an 80 per cent recovery of the Mn in the slag, 460 pounds of alloy, analyzing 78.5 per cent Mn, 14.4 per cent Fe, 1.0 per cent Si, 6.0 per cent C, and 0.15 per cent P, and about 550 pounds of waste slag.

The recovery of the Mn in the final alloy would be 64 per cent of that in the original ore. A 3,000-kw. installation would then use a 1,750-kw. furnace for the first operation and one of 1,250-kw. for the second.

In run 55, 190 pounds of slag chosen from the slags made in run 54 and in another similar run were mixed, the composition of the mixed slags being 41.0 per cent Mn, 15 per cent SiO_2, 3.4 per cent Fe, and 0.06 per cent P.

Results of run 55.

Heat.	Charge.		Charcoal.	Coke.	Mill scale.	Steel scrap.
	Mixed slags.	$CaCO_3$				
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*
a	100	10	18	....	2.5	....
b	90	9	8	8	....	1.5

Heat.	Time.	Kw.-hours used.	Alloy.	Analysis of alloy.			Slag.	Analysis of slag.	
				Mn.	Si.	P.		Mn.	P.
	H. min.		*Lbs.*	*Per ct.*	*Per ct.*	*Per ct.*	*Lbs.*	*Per ct.*	*Per ct.*
a	2 50	176	43.5	74.2	3.2	0.09	48.2	11.8	(a)
b	2 10	106	41.0	74.5	5.7	.10	45.0	5.4	(a)
Total	5 0	282	84.5	......	......	......	83.2	......	......
Average	....	......	......	74.3	4.4	.10	......	8.7	......

DISTRIBUTION.

	Mn.	P.
	Per cent.	*Per cent.*
Metal	82½	75
Slag	6½	....
Loss	11	....

a Not determined.

The results of this run checked the assumptions made as to recovery and power comsumption in smelting the slag.

If the ore were high in phosphorus, the first furnace would be so operated as to throw as little manganese into the alloy as possible, and produce a pig iron with a manganese content of about 10 per cent and a phosphorus content according to that in the ore. Such pig iron would doubtless find use when mixed with other pig of suitable composition and should have some value. If the ore is low enough in phosphorus, a little more reducer would be used in the first furnace so as to cause enough manganese to be thrown down to form a standard spiegel. If too little iron in proportion to the manganese was left in the slag for the second smelting, steel scrap or ore low in manganese but high in iron would be added, so that the products from a low phosphorus ore could be speigeleisen and ferromanganese.

COMPILATION OF DATA.

The probable performance of several types of domestic manganese ore under electric smelting, compiled from the data available on present practice, and from the results of experimental work, is given in Table 31 following.

The electrode consumption per ton of product has been calculated on the basis of kw.-hours required per ton of product, taking as standard the normal commercial consumption of 175 pounds of electrode per ton of 80 per cent ferromanganese produced from an ore containing 40 per cent manganese and 20 per cent silica, and using 5,500 kw.-hours. In other words, the electrode consumption per 1,000 kw.-hours used is 31 pounds. The actual consumption of electrodes, not including stub ends in the experimental runs, varied from 6 to 10 pounds per 450 kw.-hours used, which is of the same general order of magnitude as the figures assumed.

In Table 32 an attempt has been made to calculate costs of operation for the eight ores represented in Table 31, on the assumption that the ores are priced as follows: Ores 1 and 2 in accordance with the War Industries Board's schedule, ores 3 and 4 at $15 per ton, ore 5 at $7.50 per ton, ores 6 and 7 at $7 per ton, and ore 8 at $8 per ton. Limestone is assumed to cost 0.2 cent a pound, fluorspar 3 cents a pound, coke 0.5 cent a pound, charcoal 1½ cents a pound, steel scrap 2 cents a pound, electrodes 15 cents a pound, and power 0.5 cent per kw.-hour used. Selling prices are calculated at $285.80 per long ton of 80 per cent ferromanganese (equivalent to $250 per long ton of 70 per cent ferromanganese), and $150 per long ton of 50 per cent ferrosilicon; and the manganese and silicon contents in a silicomanganese alloy are calculated as having the same value as in the separate ferro alloys. The alloy of ore 5, with 0.60 per cent phosphorus, is figured as worth $260 per equivalent ton of 80 per cent ferro, while that of ore 7, with 0.75 per cent phosphorus, is figured as worth $235 per equivalent ton. All these figures are assumed, and would vary widely according to the situation of the plant. They are included merely to give a rough idea of the costs. Peace-time figures would greatly alter the costs and profits.

Lyon, Keeney, and Cullen[a] give the following prices per metric ton in Germany, f. o. b. Louisberg, January 1, 1913:

50 per cent ferrosilicon	$77.55
80 per cent ferromanganese	68.00
Silicomanganese (68 to 75 per cent Mn, 20 to 25 per cent Si)	106.50

It is seen that the price paid for contained manganese and silicon in the silicomanganese is higher than that paid for the same amount of manganese and silicon in ferro alloys.

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[a] Lyon, D. A., Keeney, R. M., and Cullen, J. F., work cited, p. 140.

The greater daily output per furnace when using the higher grade ores makes their use more profitable to the electric-smelting plant. On the assumptions made, the practice of smelting ores that can be run without a slag to silicomanganese shows the largest profit as well as the highest recovery of manganese from the ore.

The figures for ores 1 to 4 have been calculated on the basis of commercial practice, and for ores 5 to 8 on the basis of the experimental work in which expensive fluorspar and charcoal was used. Experience in large-scale operation would probably allow the substitution of some limestone for fluorspar and some coke for charcoal. No experiments to test this possibility were made, as the work was aimed to show the metallurgical rather than economic possibilities, and as such tests on a laboratory scale would have to be checked by large-scale tests before accurate information would be obtained.

According to Willcox,[a] Germany made silicomanganese from blast-furnace slags, and ferro or silico from low-grade ores during her manganese shortage.

The Taylor-Wharton Steel Co., High Bridge, N. J., which makes manganese steel, had a slag from manganese steel running 35 to 42 per cent silica and 31 to 43 per cent manganese.[b] The company had some of this slag electrically smelted into an alloy of 47 per cent manganese and 20 per cent silicon, which was used successfully in the manufacture of steel. The cost of smelting was too high to make the use of the alloy economically desirable at the peace-time prices of ferro then prevailing.

Tables 31 and 32 follow on next pages.

CONCLUSION AS TO ELECTRIC SMELTING.

The conclusion to be drawn from this investigation is that although the electric smelting of manganiferous slags and low-grade domestic ores is unlikely to be profitable at times of normal costs and prices, such smelting is metallurgically possible, and could be done profitably in times of high prices.

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[a] Willcox, F. H., The significance of manganese in American steel metallurgy: Trans. Am. Inst. Min. Eng. Vol. 56, February, 1917, pp. 412–420.

[b] Personal communication.

TABLE 30.—*Data showing probable results of electric smelting of domestic manganese ores.*

[Based on results of experiments and on data available from present practice.]

Ore No.	Kind of ore.	Results of electric smelting of domestic manganese ore.								Analysis of alloy made.					Analysis of slag made.		
		Analysis of ore.						Loss on ignition.	Recovery of Mn in ore as alloy.	Mn.	Fe.	Si.	C.	P.	Mn.	CaO.	SiO$_2$.
		Mn.	Fe.	SiO$_2$.	CaO+MgO.	Al$_2$O$_3$	P.										
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
1	Mixed Colorado and Utah	40.0	1.0	13.0	5.0	5.0	0.15	13.0	75	80.0	12.8	1.0	6.0	0.20	12	35	30
2	Washed Philipsburg	42.0	1.3	20.0	6.5	4.0	.10	12.0	72	80.0	11.8	2.0	6.0	.18	......	......	30
3	Mixed California ore plus quartz	30.0	1.0	35.0	.5	.5	.11	12.0	95	53.0	23.8	23.0	.6	.18	......	......	......
4	Same as No. 3 without steel scrap	30.0	1.0	35.0	.5	.5	.11	12.0	95	68.0	2.0	29.5	.3	.23	......	......	......
5	Philipsburg screenings	26.4	2.5	36.4	6.5	2.0	.21	14.5	70	71.0	11.4	15.5	1.5	.60	11	25	55
6	Philipsburg lump—one operation	24.0	1.6	38.0	7.5	1.6	.37	12.8	70	73.0	1.7	17.0	1.5	.75	12	25	55
7	Philipsburg lump—two operations	24.0	1.6	38.0	7.5	1.6	.37	12.8	a4	50.0	41.0	.5	6.0	2.50	23	......	......
									60	76.0	4.5	18.3	1.0	.20	10	25	55
8	Sultana lump—two operations	29.0	24.2	4.5	1.5	.3	.15	12.3	10	13.0	84.5	.3	2.0	.20	42	......	......
									64	78.5	14.4	1.0	6.0	.15	10	40	30

Ore No.	Kind of ore.	Charges required to produce 1 long ton of 80 per cent ferro or its equivalent in Mn (1,792 pounds).						Slag made per 1,792 pounds of Mn	K.W.H. used per 1,792 pounds of Mn	K.W.H. per long ton of the alloy made.	Carbon electrodes used.		Output per day per 3,000 k.v.a. connected load; 90 per cent power factor, 90 per cent load factor assumed.		Silicon output per day, calculated to equivalent 50 per cent Si ferrosilicon.	Long tons of high-Mn high-P pig iron.
		Ore or mixture.	CaO_3.	CaF_2.	Coke.	Charcoal.	Steel scrap.				Per 1,792 pounds of Mn made.	Per long ton of alloy made.	Alloy.	Equivalent 80 per cent ferro.		
		Lbs.	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*			*Lbs.*	*Lbs.*	*Long tons.*	*Long tons.*	*Long tons.*	
1	Mixed Colorado and Utah	6,000	1,100	....	1,800	....	230	2,600	5,000	5,000	155	155	11.7	11.7	....	....
2	Washed Philipsburg	6,000	2,100	....	1,900	....	190	4,000	5,500	5,500	175	175	10.6	10.6	....	....
3	Mixed California ore plus quartz	6,300	....	....	2,400	....	700	....	8,750	5,800	270	180	10.0	6.6	5.8	....
4	Same as No. 3 without steel scrap	6,300	....	....	2,400	....	....	....	8,700	7,400	215	185	7.9	6.7	5.8	....
5	Philipsburg screenings	10,250	....	1,450	1,200	1,200	....	6,000	10,000	9,000	310	280	6.5	5.8	2.0	....
6	Philipsburg lump—one operation	10,700	....	1,500	1,200	1,200	....	6,200	11,500	10,500	355	325	5.5	5.0	1.9	....
7	Philipsburg lump—two operations	12,500	....	940	250	250	....	[b]11,000	[c]5,500	....	390	365	4.5	4.3	1.65	....
		[d]11,000	....	860	950	950	....	6,000	[d]7,000	....	....	....	....	....	....	7.1
				1,800	1,200	1,200			12,500	11,800						
8	Sultana lump—two operations	10,000	1,000	....	500	500	....	[b]5,500	[e]5,000	....	265	260	6.9	6.8	....	....
		[b]5,500	250	....	825	825	....	2,750	[f]3,500	....	....	....	....	....	....	....
			1,250		1,325	1,325			8,500	8,350						

[a] 250 pounds of waste metal.

[b] Slag A.

[c] In 1,350-k. w. furnace.

[d] In 1,650-k. w. furnace.

[e] In 1,750-k. w. furnace.

[f] In 1,250-k. w. furnace.

TABLE 31.—*Data showing calculated costs of electric smelting of domestic manganese ores.*

Ore No. (See Table 31)	1	2	3	4	5	6	7	8
Kind of ore	Colorado and Utah.	Washed Phillipsburg.	California ore and quartz.	Same as 3 without steel scrap.	Philipsburg screenings.	Philipsburg lump—one operation.	Philipsburg lump—two operations.	Sultana lump—two operations.
Cost of ore per ton	$38.30	$36.25	$15.00	.....	$7.50	$7.00	$7.00	$8.00
Cost per long ton of 80 per cent ferro or equivalent Mn (1,792 pounds):								
Ore	102.50	97.50	42.00	$42.00	34.30	34.40	39.00	35.80
CaCO$_3$	2.20	4.20	.....	.....	.....	.....	.....	.....
CaF$_2$	.....	.....	.....	.....	43.50	45.00	54.00	.....
Coke	9.00	9.50	12.00	12.00	6.00	6.00	6.00	6.65
Charcoal	.....	.....	.....	.....	18.00	18.00	18.00	19.90
Steel scrap	4.60	3.80	14.00	.....	.....	.....	.....	.....
Power	25.00	27.50	43.75	43.50	50.00	57.50	62.50	42.50
Electrodes	23.15	26.20	40.50	40.50	46.50	54.50	58.50	40.00
Total	166.45	168.70	151.25	138.00	198.30	215.30	238.00	147.35
Value of product:								
On basis of long tons of 80 per cent ferromanganese at $285[a]	(1) 285.00	(1) 285.00	(1) 285.00	(1) 285.00	(1) [b]260.00	(1) [c]235.00	(1) 285.00	(1) 285.00
On basis of long tons of 50 per cent ferrosilicon at $150[a]	.....	.....	(0.88) 132.00	(0.88) 132.00	(0.35) 52.50	(0.38) 57.00	(0.38) 52.00	.....
On basis of long tons of pig iron with 10 per cent Mn at $30	.....	.....	.....	.....	.....	.....	1.05	31.50
Calculated value of alloy per 1,792 pounds of Mn contained	285.00	285.00	417.00	417.00	312.50	292.00	337.00	316.50
Difference-value of product less cost of raw materials	119.55	116.30	265.75	279.00	114.20	76.70	99.00	169.15
Difference times equivalent tons of ferromanganese made per day[a]	(11.7) 1,400.00	(10.6) 1,235.00	(6.6) 1,750.00	(6.7) 1,870.00	(5.8) 605.00	(5.0) 383.00	(4.3) 425.00	(6.8) 1,150.00
Assumed cost of superintendence, labor, overhead, interest, depreciation, etc., per day, per 3,000 k. v. a. of plant capacity	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00
Calculated profit per day per 3,000 k. v. a.	1,000.00	835.00	1,350.00	1,470.00	205.00	(loss) 17.00	25.00	750.00

[a] Figures in parentheses represent tons.
[b] Assumed penalty for 0.60 per cent P, $25 per ton of product equivalent to 80 per cent ferro manganese.
[c] Assumed penalty for 0.75 per cent P, $50 per ton of product equivalent to 80 per cent ferro manganese.

CHAPTER 11.—USE OF MANGANESE ALLOYS IN OPEN-HEARTH STEEL PRACTICE.

By Samuel L. Hoyt.

INTRODUCTORY STATEMENT.

This report presents the results of an extensive study of the use of manganese alloys in open-hearth steel practice in the United States. The magnitude of the work and the number of men who aided in one way or another make difficult acknowledgment to all who have contributed to its progress. However, some expression should be made of the hearty cooperation accorded to the members of the bureau by the various manufacturing interests. With such a sympathetic attitude, the investigation, depending so largely, as it did, upon cooperation between the steel plants and the investigators, was early assured of every possibility of ultimate success. The war conditions that rendered this investigation necessary have passed, but it is hoped that the results may prove of permanent value to the open-hearth steel industry.

PURPOSE AND SCOPE OF INVESTIGATION.

The purpose of making this investigation was to determine the extent to which domestic or low-grade manganese alloys could properly be substituted in open-hearth steel practice for high-grade alloys without materially impairing the steel production either as to quality or quantity. Moreover, it was held that such an extensive investigation of this important step in the manufacture of steel would undoubtedly yield valuable results to the steel industry as well as contribute, in no small way, toward directing future investigations in the same field.

It was recognized at the start that data bearing on the projected study should be available at individual plants, and that a compilation and digest of such results would be the logical method of approaching the proposed investigation. This consideration somewhat controlled the selection of the steel plants at which to conduct the detailed investigations.

The evolving of a definite experimental program from the statement of the general problem, considering the time element and the many and varied factors involved, was not reached without due consideration of competent metallurgical advice. After making a preliminary survey, it seemed important to determine (*a*) the conditions in open-hearth practice that would lead to a conservation of manganese, both during the working of the heat and in making the final additions; (*b*) the most satisfactory metallurgical conditions for the use of manganese in the form of low-grade or special alloys; and (*c*) the effect on the finished steel, both as to quality and "condition," of the various methods and processes studied. With these points in mind, the selection of the steel plants was made so that research work bearing on one or more of these points could be conducted.

It was decided to determine slag and metal compositions during the refining of the heat; also, the temperature was to be noted each time a sample was taken, in order to determine, if possible, the temperature effect. The "recovery" of manganese was to be determined from the residual and final manganese contents and the weight of the metal. To this end a sample of the finished steel was taken during teeming. By taking three such samples, one at the beginning, one toward the middle, and one at the end of teeming, tests for uniformity were possible. This practice was generally observed throughout the investigation. The data obtained were also supplemented by the plant records covering given heats as well as by personal observation during refining, pouring, and teeming.

When planning steps that should be taken to determine the quality and "condition" of the steel, it was found that no definite and well-proved method was available. True, the open-hearth melter knows whether his heat is in proper condition, but what was needed was a quantitative estimate of "condition." Without attempting to discuss the physical chemistry of a heat of molten steel, it may be said that the condition of the heat must depend, aside from the temperature, upon the presence in the steel in those substances that affect the "condition." Of these there are two kinds: (1) Substances that promote "openness," or the gases, which again may be classified as (*a*) gases that are products of chemical reactions, being, in so far as we know, CO and possibly CO_2, and (*b*) gases that are absorbed from the furnace gases, such as H, N, CO, and CO_2; and (2) substances that promote "soundness," such as the reducing and solidifying agents, C, Mn, Si, and Al.

In general it is held that Mn, Si, and Al inhibit the chemical reactions producing CO by reducing (or partly reducing) FeO, the principal constituent that produces the reactions. In this statement only the metal bath is considered and the FeO, and not Fe_2O_3, is assumed to be in solution in the steel. According to this idea, reducing action on a slag containing Fe_2O_3 would produce FeO, part of which would enter the steel to react later with C, Mn, ıd other reducing agents present. The reduction of FeO, then, is the principal means of "settling" the liquid steel, and it is for this reason that Mn is added in the final steps. It is also held that Si and Al produce solidity in the finished steel, aside from reducing FeO and CO, either by keeping the gases H, N, etc., in solid solution, or by preventing the dissociation of the compounds of those elements and iron.

The obvious procedure to get a quantitative estimate of the "condition" of the steel, considering both the behavior of the molten metal and the character of the ingot, would be to determine the amounts of the constituents in each of these two groups and to weigh one set against the other. Even this procedure would not, at present, lead to results that could be interpreted with entire confidence, even though there were no uncertainties in the analytical methods, because we do not know the quantitative effect of each constituent, either by itself or when associated with other constituents in varying amounts. In view of this lack of fundamental data, it was decided to make the analyses and use the results in a qualitative way, at least, to compare the different practices investigated.

THE FUNCTIONS OF MANGANESE.

During such a critical period as that now passed, the question might be raised as to the possibility of eliminating manganese from steel making. This point was duly considered but it was at once held that the use of manganese is not merely an expedient, for which some substitute might readily be had, but is rather one of the basic requirements of successful practice in working steel. It is quite true that in many instances the actual amount of manganese used in a heat of steel is greater than purely metallurgical considerations demand, and any excess could well be considered as so much wasted.

It may be well to review briefly the important functions of manganese as they bear directly on both of the points mentioned above. The first function of manganese, broadly considered, is to refine and "settle" the molten bath of steel. The aim here is to put the metal in a proper condition for pouring, and to produce ingots (or castings) of the desired quality and texture. Manganese is not the most efficient element that can be used for this purpose, calculated from the heat of combustion of the element to its oxide, but is without doubt the most satisfactory because of the excellent condition (freedom from objectionable foreign inclusions) in which it leaves the bath.

The proportion of manganese theoretically required for this operation might possibly be calculated from the amount of oxygen converted from the active form, FeO, to the inactive form, MnO. Assuming an oxygen content of 0.075 per cent in the unsettled steel and of 0.015 per cent in the finished steel (oxygen by the Ledebur method), the amount of manganese used in this way would be 0.2

per cent. The writer is informed by the Bureau of Standards that such a calculation is premature, owing to lack of knowledge on the subject of "deoxidation" and the faultiness of the Ledebur determinations. However it would seem to the writer, from the work done at the Bureau of Standards,[a] that the amount of oxygen determined is the amount present as FeO (active form), subject possibly to an error due to partial reduction of CO during the determination. At any rate, the above is advanced as at least the first approximation of the amount of manganese required simply for destroying the ferrous oxide present in the bath. The amount of manganese required naturally would vary with the condition of the bath and, in order to insure efficient "deoxidation," would be somewhat in excess of the calculated amount. A well-made heat of steel would probably not require more than 0.35 per cent Mn.

Manganese is also desirable in steel to improve the rolling properties, in which capacity it appears to serve a dual purpose. First of all, manganese deoxidizes and refines the molten steel in such a way as to give ingots of the desired texture without robbing the steel of its hot-working properties. Thus, ingots may be rolled into finished shape, without the formation of excessive fissuring or surface defects. Other reducing agents, such as aluminum and silicon, are prone to leave the metal in poor condition for rolling and forging. They eliminate one cause of hot shortness—iron oxide—but fail to convert the sulphur into a harmless form, as manganese does, and leave behind their highly refractory oxides, both of which tend to produce poor rolling qualities. Secondly, manganese, by retarding the rate of coalescence or grain growth, renders steel less sensitive to the effects of the high temperatures used in rolling and is supposed to promote plasticity, at least in ordinary steels, at rolling temperatures. Silicon and aluminum, on the other hand, increase, rather than decrease, the grain size of steel. The proportion of manganese required in this capacity probably does not exceed 0.35 per cent in well-made steel.

Finally, manganese is desired in the finished steel to produce certain physical or mechanical properties or to make the steel more amenable to subsequent heat treatment.

The foregoing discussion indicates that manganese is an important factor in the steel industry. Of course, material such as "American ingot iron" can be successfully rolled, even though no manganese be added, but requires greater time and care.

It is of interest to note that manganese, coming in the periodic system between iron and the strengthening elements on one side and the hardening elements on the other, has the dual function of strengthening and hardening steel, which is not possessed by any other element.

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[a] Cain, J. R., and Pettijohn, Earl. A critical study of the Ledebur method for determining oxygen in iron and steel: Bureau of Standards Technologic Paper 118, 1919, 33 pp.

Manganese conservation would best be obtained by eliminating the manganese specification except where the amount of manganese present in the finished steel has some definite bearing on the properties of heat treatment of the steel. In other words, whenever only casting and rolling-mill practice (plant problems) are involved, the steel man should be allowed to exercise his own judgment as to the amount of manganese that should be used to give the most satisfactory and economical practice, and the finish and quality of the product should be controlled by adequate inspection. On the basis of Ellicott's figures,[a] by reducing the manganese requirements by 0.2 per cent in making plates and shapes and other low-carbon steel (estimated production, 21,350,000 tons), 54,000 tons of 80 per cent ferromanganese could be saved.

RECOMMENDATIONS FOR THE UTILIZATION OF DOMESTIC ALLOYS.

The investigation here reported indicated that three practices for utilizing domestic alloys in open-hearth steel practice seem to commend themselves above the others. These are as follows (but not in the order of their importance): (1) The use of a "molten spiegel mixture" for deoxidation and recarburization; (2) the practice of melting and refining the steel bath so as to insure a comparatively high residual manganese content, say, 0.3 per cent Mn; (3) the use of manganese alloys containing silicon. In selecting plants for investigating these practices two points were kept in mind. The plant should have either "ordinary" practice, for the sake of comparison, or else one of the three just mentioned, and the product or kind of steel made should be representative of the larger tonnages, such as shell steel, plates, or sections.

"MOLTEN SPIEGEL MIXTURE" PRACTICE.

The practice has been adopted at a few plants of combining in one operation both recarburization and deoxidation by using a mixture of pig iron and spiegel that has been premelted in a cupola. This "molten spiegel mixture" contains 5 to 11 per cent metallic manganese, 4 per cent carbon, and the desired amount of silicon, and is added to the ladle during the tapping in such a way as to cause a thorough and uniform mixture of the two streams.

The principal advantages of interest here, not considering questions of plant and operating economy, are as follows: 1. A low-grade or domestic alloy can be used in the preparation of the "mixture."

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[a] Ellicott, C. R. Manganese conservation in steel making: Iron Age, vol. 101, June 6, 1918, pp. 1484-1485.

2. The deoxidation is accomplished by means of a dilute solution, with a consequent increase (on theoretical grounds) of the efficiency of the deoxidizer. This point will receive further consideration. 3. The deoxidizer is added in the molten state, insuring certain attendant advantages, which will also be considered at greater length. 4. A special advantage, if a large steel output is desired, is that the amount of the recarburizer is comparatively large and the capacity of the plant is materially (and economically) increased thereby. There is some question as to the propriety of including this advantage as peculiar to this particular practice. The use of pig iron as a recarburizer may be accomplished in other ways with the same economy and increase in plant capacity. 5. Another advantage would seem to the writer to be as follows: As compared with the results in the usual practice of adding carbon and manganese, there should be less likelihood of missing a heat.

This practice, at least at the plant visited, and it is understood to be the same elsewhere, is limited to the manufacture of the high-carbon steels or those running 0.30 per cent or more of carbon. To make steels with 0.20 per cent carbon would require working the bath until the carbon content was about 0.10 per cent, and the molten mixture added would have to contain about 20 per cent manganese (spiegel). The amount of the addition would be reduced from 13,000 pounds to about 4,000 pounds, which would mean that some of the advantages just enumerated would be lessened, and, with the increased loss of manganese the practice would probably not be commercially feasible. However, when the other alternative—the use of ferromanganese, either solid or liquid—is considered, the practice of premelting spiegel in the cupola seems commendable, on grounds to be considered later. In the event of undue shortage of high-grade ferromanganese the practice would doubtless offer a ready solution of the problem of using domestic alloys in making steel for shapes, plates, etc. Against the increased cost of production, as compared with the cost of cold ferromanganese practice, there would be the greater uniformity of product and more uniform practice as an offset.

HIGH RESIDUAL MANGANESE PRACTICE.

At certain plants the practice of preferential oxidation and elimination of carbon and phosphorus has been developed, the residual manganese being kept at a comparatively high value, say, 0.25 to 0.30 per cent, as compared with 0.10 per cent manganese for a final carbon content of 0.10 per cent. This is accomplished, broadly speaking, (*a*) by rapidly removing the phosphorus and retaining it as stable calcium phosphate during the earlier and colder period of melting; (*b*) by maintaining a high finishing temperature and working the charge with a high manganese content so that the slag contains about 8 per cent manganese; and (*c*) by increasing the lime content of the slag to about 47 per cent as a minimum.

This process possesses undoubted advantages, but they are such that they are probably best appreciated by plants in which the process has been developed and where it is now in operation on a sound commercial basis. First of all it may be stated that the practice, correctly applied, leads to the production of high-grade and uniform steel, which in itself means increased rolling-mill output, fewer rejections, and a more ready market. This is largely due to the fact that the steel is made—where it should be made—in the furnace.

A second advantage derived from the high MnO and CaO contents of the slag, is that the final additions of manganese can be added in the furnace, with a recovery comparing favorably with that of ladle additions. A third advantage is that the same pig iron used for the charge, and containing appreciably more manganese than ordinary basic iron does, can be used to recarburize and partly deoxidize the bath. The rest of the manganese is added as ferromanganese. At a steel plant which operates in conjunction with a blast-furnace plant a harmonious and economical cycle of plant operations is made possible. At the same time the open-hearth slag can be resmelted in the blast furnace for the recovery of the iron and manganese and the utilization of the lime.

This practice is largely dependent upon the amount of phosphorus in the slag, for obviously it would not be worth while to recover the manganese at the expense of unduly increasing the phosphorus content of the pig iron. In this country we are fortunately situated in this respect, as there is still a large amount of ore rather low in phosphorus available. No definite figure can be given at this time as to the maximum allowable phosphorus content of the pig iron, but it is the opinion of at least one steel man who uses this process that a content of 0.6 per cent would not be excessive. Under the conditions prevailing in 1918, this practice had the additional advantages that the high-manganese pig iron could be procured by smelting domestic manganiferous iron ore and that the manganese alloy added to the furnace at the end of the heat could as well be spiegel as ferromanganese, assuming that the finished steel contains more than about 0.10 per cent carbon. There would also be certain disadvantages, particularly that the carbon content of the bath would have to be worked to a lower figure than in present practice. On account of the high cost of spiegel and the greater time required, it is doubtful whether the steel plants would substitute spiegel for ferromanganese. Another interesting point, as regards the utilizing of domestic manganiferous iron ore, is that low-silica ore could be added to the slag as a source of manganese oxide.

The high manganese content of the charge is generally obtained by using a "high-manganese" pig iron (2 to 3 per cent manganese), but may also be obtained by adding manganese ore to the slag or manganese alloys to the bath or by a combination of these methods. This point would be determined by plant economy, but it seems doubtful whether the practice would be worth while unless a high-manganese pig iron were available. The writer is informed by one blast-furnace superintendent that running the manganese up to 2 per cent does not materially affect the production, so that lower pig-iron production would not be a disadvantage in this practice. The loss of manganese by oxidation and transferrence to the slag is considerable. This loss may be kept at a minimum by increasing the basicity of the slag in CaO and FeO, which, combined with the MnO, which also acts as a base, exert the desired effect upon the manganese of the bath.

As the working of the charge progresses its temperature rises until finally with the high CaO, and particularly the high MnO content of the slag, the carbon is eliminated more rapidly than the manganese, with the result already stated, namely, the manganese can be held to about 0.3 per cent at the end of the heat. Present data indicate, unfortunately, that no material decrease in the amount of manganese required and no material increase in the recovery of manganese in the additions may be expected, so that the advantages are derived not from a decreased consumption, but from the form in which it can be added.

Data for one such heat showed that a total of 3,728 pounds of manganese was used in one form or another to produce 1,272 pounds of manganese in the finished steel—that is, 3.54 pounds was used to produce 1 pound in the steel. The manganese added in the recarburizer and as ferromanganese amounted to 1,068 pounds, of which, assuming the manganese loss to come from these two sources, 838 pounds was recovered in the finished steel, a recovery of 78.4 per cent. In this heat the ferromanganese was added to the furnace. Another more or less comparable heat selected at random, but more representative of "standard" practice, used 2,190 pounds of manganese to produce 1,200 pounds, or 1.82 pounds (as compared to 3.54 pounds) to produce 1 pound of manganese in the finished steel.

STANDARD OPEN-HEARTH PRACTICE COMPARED WITH CERTAIN OTHER PRACTICES.

Data regarding the results obtained in standard open-hearth steel practice as compared with the results when molten spiegel is used or when there is a large percentage of residual manganese will be presented in a later report after more complete analytical results have been received.

USE OF MANGANESE-SILICON ALLOYS.

The high silica content of most of our domestic manganese and manganiferous iron ores made it advisable to consider the possible use of manganese-silicon alloys in steel making in both acid and basic practice. For the purposes of the present discussion these alloys will be divided roughly into two classes—high-grade silicomanganese containing about 50 per cent manganese and 25 per cent silicon and low-grade silicospiegel with about 15 to 20 per cent silicon and 30 to 35 per cent manganese with 50 per cent iron. The manganese-silicon ratio of the first alloy is about 2 and of the second alloy 2⅓ to 1½. Each of these alloys would be made from the silicious manganese ores of California and Montana, and the low-grade alloys from the silicious manganiferous iron ores of Minnesota.

While there is nothing new about the practice of using manganese-silicon alloys in steel making, it may be well to review some of the points connected therewith.

It is understood that silicomanganese has been used fairly extensively in Europe, and in this country it was used at certain plants as standard practice until the supply was cut off by the war. Silicon is always an efficient reducing or settling agent when used in the customary small amounts, but it may or may not be desirable in the finished steel. On this account the possibility of using manganese-silicon alloys depends upon the amount of silicon that can be tolerated in the finished steel in the ingot form. In certain grades of steel, particularly in steel that must be welded, silicon should be low or practically absent. In steel for sheets and plates, which must give a good finished surface, the most efficient rolling-mill practice requires that the silicon be kept tolerably low; but it is believed that 0.10 to 0.15 per cent could be used, provided the manganese content were not too high. In forging steel, high-carbon steels, and castings, where the aim is to produce sound steel, more silicon can be used, or between 0.20 and 0.35 per cent. Of these three fields the latter is the one in which manganese-silicon alloys will find their first application. In the second field it seems quite probable that conditions (to be discussed later) will many times permit their use; but from the nature of things manganese-silicon alloys can not be used to make steels of the first group—those that must be welded.

ACID PRACTICE.

It is with considerable hesitation that the discussion of manganese-silicon alloys in open-hearth practice is approached, particularly as the controversial character of many of the points is so clearly recognized. Consequently, it may be well, at the outset, to state briefly the manner in which the writer first became interested in the possibilities of their use. A number of years ago the writer was conducting a series of experiments on the occurrence and identification of foreign inclusions in acid open-hearth steel, principally ordnance steel. In this work ferromanganese, ferrosilicon, and a mixture of ferromanganese and ferrosilicon were added to a steel sample taken shortly after "oreing"—that is, to "wild" steel—in an attempt to produce an excess of the constituent, or constituents, supposed to form as a result of the addition.

It seemed fairly clear as a result of this work that the use of silicon was apt to be dangerous, not on account of any harmful effect of the residual metallic silicon but because it produced a constituent (assumed to be SiO_2 or at least a highly refractory silicate) that was likely to remain in the ingot and produce hot shortness. Hence the idea was suggested that a manganese-silicon alloy might, and probably would, form a manganese silicate containing some ferrous oxide (a true slag) which would be fluid and would more readily coalesce into larger particles than SiO_2 would, and therefore free itself more readily from the steel. By using such an alloy it would then be possible to take full advantage of the use of silicon as a deoxidizer without suffering the usual attendant disadvantages of its use. None of the manganese-silicon alloy was available at the time so a parallel experiment could not be conducted.

As binary alloys are known to be generally more active, or powerful, than the weighted sum of the two constituents would indicate, it was also assumed that, aside from the possibility of obtaining a better separation of the insoluble products of the deoxidation process, the alloy of manganese and silicon would prove to be a more powerful reducing agent than ferromanganese and ferrosilicon used separately. On reflection, the thought occurs that manganese and silicon, reacting separately with FeO, would produce the oxides MnO and FeO or a silicate of iron. Manganese and silicon reacting as an alloy with FeO would produce a silicate of manganese, which may or may not form a double silicate with FeO. In either case we would expect to find the advantage in favor of the manganese-silicon alloy.

The relative weights of the silicomanganese and of the mixture of the ferromanganese and ferrosilicon will be considered at another place.

Another point of great technical importance is the percentage recovery of manganese when added as silicomanganese and as ferromanganese along with ferrosilicon. It should be stated that a 100 per cent recovery, based on the present theory of "deoxidation," is hardly possible, nor is it desirable. Such recovery would mean retention of the deoxidation products, to be determined later as metallic manganese and silicon. A method of addition that would lead to satisfactory deoxidation and yet would eliminate the loss due to admixture with the slag, volatilization, etc., and could be accomplished with the minimum amount of manganese, would be very desirable because it would lead to both conservation of manganese and uniformity of composition of the steel. Conservation of manganese would be given by the actual percentage recovery, and uniformity of composition would be assumed by the constancy of the percentage recovery.

Fortunately the writer was able to examine records of heats made with silicomanganese covering a period of several years, from which some fairly satisfactory conclusions may be drawn bearing on these points. During this time when there were periods when the silicomanganese alloy was not available and a mixture of ferromanganese and ferrosilicon had to be substituted. Thus, direct comparison of these two methods of deoxidation was afforded. Certain results taken from the heat records, and believed to be typical, are given in Table 33 following. Obviously, figures showing the variation in heat composition and the average manganese recovery of several years' practice can not be given in this table. The records themselves clearly show greater uniformity for the silicomanganese heats.

TABLE 33.—*Comparative results obtained with silicomanganese and with a mixture of ferromanganese and ferrosilicon.*[a]

Heat.	FeSi.	FeMn.	SiMn.	Total charge.	C.	Mn.	Si.	Mn. added.	Mn. recovered.	
	Lb.	*Lb.*	*Lb.*	*Lb.*	*Per ct.*	*Per ct.*	*Per ct.*	*Lb.*	*Lb.*	*Per ct.*
A	160	300	....	31,160	0.26	0.56	0.294	240	174	72.5
B	215	400	....	40,765	.21	.57	.306	320	232	72.5
C	160	310	....	30,620	.32	.63	.312	248	193	77.8
D	160	300	....	30,910	.27	.70	.318	240	216	90.0
E	215	400	....	40,865	.24	.72	.312	320	294	91.8
F	....	40	470	41,010	.22	.58	.308	281	238	84.7
G	....	35	350	30,685	.26	.60	.302	214	184	86.0
H	....	40	470	40,960	.21	.64	.310	281	261	93.0
I	....	40	470	30,510	.21	.66	.303	281	267	95.0
J	....	40	420	36,760	.24	.68	.310	255	250	98.0

[a] Approximate compositions: Silicomanganese, Mn 53 per cent, Si 20 per cent; ferromanganese, Mn 80 per cent; ferrosilicon, Si 50 per cent. Residual manganese was neglected in calculating recoveries.

It can hardly be claimed that these figures, or the three years' records which they represent with reasonable accuracy, furnish a truly scientific basis for comparison of the two alternate practices, but they do show rather convincingly that the same results (Mn and Si contents of the finished steel), by using silicomanganese can be obtained with consistently smaller amounts of both manganese and silicon, as compared with the combination of ferromanganese and ferrosilicon. In addition there is the advantage of more uniform practice, which in itself would warrant smaller additions. The weights of the additions favor the silicomanganese; thus in heats A, C, and D, 460 pounds was added, as compared with 385 pounds for heat G, and in heats B and E, 615 pounds was added, as compared with 510 pounds in heats F, H, and I. The low carbon content of the silicomanganese may or may not be a material advantage, but is in favor of the single-alloy addition because the carbon need not be worked as low and there seems to be less danger of missing the desired carbon content.

ELECTRIC-FURNACE PRACTICE.

No information is available to the writer bearing on the use of manganese-silicon alloys in electric-furnace practice, but we may at least consider such practice on the basis of the known behavior of such alloys. Considering acid casting practice first, there seems to be no reasonable doubt that either silicomanganese or silicospiegel could be at once substituted for ferromanganese and ferrosilicon. Inasmuch as the usual aim is to make high-grade castings, the manganese-silicon alloys would appear to have the distinct advantage of making sounder and cleaner steel. Silicospiegel, aside from possessing the theoretical advantage of being diluted with iron, could be more readily prepared with the correct manganese-silicon ratio so as to eliminate the use of an additional alloy. In this practice the advantage of greater dilution need not carry with it the disadvantage of increased weight on account of the higher temperature of the electric furnace. The uncertainty as to the relative behavior of the manganese-silicon alloys as compared with that of the ferro-alloys, and the relative efficiencies of low-grade and of high-grade alloys, as well as the importance of this step in the manufacture of steel, suggest the advisability of conducting a definite research to settle such points. It would seem that there is no better place for such a research than in this particular industry.

In basic electric-furnace practice the manganese-silicon alloys, on the same grounds, could likewise be utilized, particularly as the attempt is always to produce sound and clean ingots. However, in this practice, ferrosilicon is used as a reducing agent along with coke, and hence the operator would probably not see any advantage in changing his practice in favor of the manganese-silicon alloys.

BASIC OPEN-HEARTH PRACTICE.

The amount of information available on the use of silicomanganese in basic open-hearth practice is meager, but it can be said that silicomanganese can be used, probably with as satisfactory results as with ferromanganese and ferrosilicon. Through the cooperation of one steel plant the writer was able to follow two shell-steel heats made with silicomanganese which was added to the ladle. The results of the second of these heats are given here to show what was done. To 11,100 pounds of molten pig iron in the ladle was tapped 122,340 pounds (estimated) of steel analyzing 0.09 per cent C, 0.12 per cent P. 0.15 per cent Mn. [illegible] per cent S [illegible] cent Si. At the same time [illegible] pounds of silicomanganese [illegible] cent Mn., 300 pounds of 7[illegible] per cent ferromanganese [illegible] aluminum, and 50 pounds of coal were added to the ladle [illegible] was in excellent condition and the ingots had smooth [illegible] displayed no superficial action or evolution of gas. The [illegible] was 0.40 per cent C, 0.028 per cent P, 0.5[illegible] per cent [illegible] per cent S, and 0.21 per cent Si. The recovery of manganese [illegible] the entire loss to come from the alloy added, was [illegible] recovery of silicon was 65.1 per cent. Only 24 pounds [illegible] 5 per cent of the total was lost. In the first heat, which was [illegible] to be more highly oxidized, the recovery of silicon was [illegible] per cent while the recovery of manganese was only [illegible] These results indicate that silicon protects manganese in [illegible] heats.

with 510 pounds in heats F, H, and I. The low carbon content of the silicomanganese may or may not be a material advantage, but is in favor of the single-alloy addition because the carbon need not be worked as low and there seems to be less danger of missing the desired carbon content.

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Bulletin 178C

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, Secretary
BUREAU OF MINES
VAN. H. MANNING, Director

PETROLEUM INVESTIGATIONS AND PRODUCTION OF HELIUM

ADVANCE CHAPTER FROM BULLETIN 178
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BY

VAN. H. MANNING

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PETROLEUM INVESTIGATIONS AND PRODUCTION OF HELIUM.

PETROLEUM INVESTIGATIONS.

The work of the petroleum division of the Bureau of Mines during the period of the war was under the supervision of Chester Naramore, chief petroleum technologist. In its petroleum investigations the bureau cooperated freely with other Government bureaus and acted as a source of information on petroleum matters.

COOPERATIVE WORK ON INTERALLIED PETROLEUM COMMISSION.

On July 15, 1918, Mr. Naramore and W. E. Perdew, chemical engineer of the bureau, left for London to act as representatives of the United States Fuel Administration and the United States Shipping Board on the Inter-allied Petroleum Commission. Numerous conferences were held with representatives from France, England, and Italy on specifications and requirements of the various allied Governments for petroleum products, and for this purpose meetings were held in London, Paris, and Rome.

EXAMINATION OF PETROLEUM FACILITIES IN FRANCE.

The storage facilities and the transportation system in France were studied, and recommendations based on the information gained were made to the director of the oil division of the Fuel Administration and to the chairman of the Shipping Board.

Refinery plants in France were inspected, as well as filling stations and shipping facilities from ports in France and Italy to the various fronts and the petroleum base of the American Expeditionary Force at Romorantin, France. At the request of Ambassador Sharp, a report was prepared on the advisability of developing the petroleum resources in France.

AIRPLANE FUELS.

Conferences were held with the various officials of the American Expeditionary Force regarding airplane fuels and lubricants. A large amount of data was collected on the tests conducted by the French and British Governments on different grades of gasoline and on special blends for airplane fuel. Various French laboratories were inspected to ascertain the methods used in testing petroleum products for airplane engines.

STUDY OF OIL-SHALE INDUSTRY IN SCOTLAND.

The oil-shale industry in Scotland was studied with a view to determining the practicability of applying the methods used there to the treatment of oil shales in the United States.

SPECIFICATIONS FOR MOTOR FUELS.

The petroleum division assisted several branches of the United States Government in drawing up specifications for the purchase of various grades of motor fuel. Specifications were written for the Panama Canal and for both motor and airplane gasoline, which were adopted by the Panama Canal Commission.

Specifications for three grades of airplane gasoline for the use of the United States Government were adopted at a conference held May 30, 1918, at the office of Mark L. Requa, director of the oil division of the Fuel Administration. At this conference Mr. Naramore and Mr. Perdew represented the Bureau of Mines. Largely through the efforts of the petroleum division, the Atlantic Refining Co. agreed to make five drums of fighting gasoline, the first gasoline of that grade to be made in this country, for tests at the Bureau of Standards. Through its experience in making this sample, the Atlantic company was enabled to manufacture fighting gasoline on a large scale at a later date. This company is the only one in the United States that has attempted to make gasoline that would meet the specifications for the fighting grade.

In August, 1918, an Executive order authorized the establishment of a committee on the standardization of specifications for petroleum products. The duties of this committee were to standardize specifications for all petroleum products used by Government departments and to make suggestions on specifications for the allied governments. Shortly after the organization of this committee, a technical subcommittee was appointed to study existing specifications and to make recommendations to the general committee. The Bureau of Mines was represented on the general committee by C. H. Beal, petroleum technologist, and on the subcommittee by H. H. Hill, assistant chemist. Later Mr. Beal was transferred to San Francisco and Mr. Hill represented the bureau on both committees. Also Mr. Hill acted in an advisory capacity to the Inter-Allied Conference at its meetings, held in this country, on specifications.

of fuel oil, kerosene, long-time burning oil; signal oil, and "mineral seal oil," and is considering specifications for lubricants, particularly those used by the Railroad Administration.

The investigations relating to specifications for petroleum products were conducted by Messrs. Naramore, Perdew, Hill, and Beal.

INVESTIGATIONS OF AIRPLANE-MOTOR FUELS.

About August 2, 1917, the bureau arranged to cooperate with the Aviation Section of the Signal Corps in studying fuels for airplane engines. Information collected through visits to a number of aviation fields, by conferences with organizations testing airplane motors, and by correspondence indicated that no reliable data were available regarding the proper fuel for aviation motors. This work was done by E. W. Dean, petroleum chemist.

TESTS WITH DIFFERENT FUELS AND ENGINES.

The bureau arranged with the engineers at Langley field to make some actual flying tests to ascertain the advantages of certain grades of gasoline. Flying tests with Curtis training planes, indicated that plain "motor" gasoline gave as good results as a "high test" Pennsylvania product. The Italian fliers at Langley reported, however, that their engines tended to overheat when "motor" gasoline was used. The tests were made by C. Netzen, chemical engineer, and E. W. Dean.

After it was seen that the problem was what increase in efficiency would be possible through improved design if the fuel were better, an arrangement was made with the officials of the McCook aviation field and with the reseach organization of the Dayton-Wright Airplane Co. to cooperate in thorough tests of different engines and gasolines. This work included tests with a small air-cooled high-compression engine, tests with a single-cylinder high-compression Liberty engine, the preparation of special fuels (including cyclohexane), flying tests at the Dayton-Wright and Wilbur Wright fields with De Haviland airplanes equipped with Liberty motors, and tests in the altitude chamber of the Bureau of Standards.

Many interesting and important facts were learned. The general conclusion was that a mixture of 70 per cent cyclohexane and 30 per cent benzol (so-called "hecter") seemed the most desirable for use in fighting airplanes. This fuel permitted the use of engines having compression ratios as high as 7.5 to 1, which applied to the Liberty motor would develop perhaps 10 per cent more power.

The cessation of hostilities prevented the development of commercial cyclohexane plants. Flying tests at the Wilbur Wright field, completed after the signing of the armistice, showed that the use of a high-compression motor with cyclohexane as fuel gave a standard De Haviland No. 4 airplane a full thousand feet extra of "ceiling" (maximum flying altitude).

The investigation of aviation gasoline was conducted chiefly by Clarence Netzen and J. P. Smootz, assistant petroleum technologists.

COOPERATIVE TESTS WITH BUREAU OF STANDARDS.

It was realized soon after this country entered the war that ordinary motor gasoline was not suitable for use in airplanes, particularly at high altitudes. The Bureau of Standards constructed an altitude chamber for testing airplane engines under conditions similar to those encountered at various altitudes from sea level to 30,000 feet. Different types of fuels were tested to determine which were most satisfactory at high altitudes. The Bureau of Mines cooperated by collecting samples of various grades of gasoline, special fuel mixtures, and close-cut gasoline. In April and May, 1918, arrangements were made with the Atlantic Refining Co. and the Union Oil Co. of California to make special close-cut gasoline with a boiling range of 10° C. if possible. A special fuel was made from absolute alcohol and gasoline, the absolute alcohol and a blend of gasoline and benzol being prepared in the Washington laboratory of the Bureau of Mines. All the fuels. about 100 samples, tested by the Bureau of Standards in airplane engines were submitted to distillation tests in that laboratory. W. E. Perdew collected the samples; C. P. Bowie and M. J. Gavin, assistant refinery engineers, assisted in obtaining gasoline cuts from the Union Oil Co. of California.

ADVICE ON GASOLINE PROBLEMS.

Considerable time was spent in advising the oil division of the Fuel Administration on problems particularly relating to tests and manufacture of gasoline. At the request of Mr. Requa, director of the oil division, a report was prepared on the potential production of airplane gasoline in the United States. This report, made by H. H. Hill with the assistance of E. W. Dean and J. P. Smootz, showed that the maximum estimated requirement could be met if the use of gasoline for motor cars was restricted.

SUPPLEMENTARY INVESTIGATIONS IN CONNECTION WITH AIRPLANE FUELS.

In some airplane engine tests at the McCook field aluminum pistons were seriously corroded. Investigation showed that the corrosion was not caused by impurities in the fuel and could be con-

Two samples of German airplane gasoline, analyzed at the Pittsburgh laboratory of the bureau by H. H. Hill, proved to be petroleum gasoline, carefully refined.

Samples of gasoline from Borneo and from Sumatra were found to be suited for aviation fuel. The analyses were made by Mr. Hill and by N. A. C. Smith, assistant petroleum chemist.

INSPECTION OF OVERSEAS SHIPMENTS OF GASOLINE.

In the latter part of July, 1918, a report was received in this country that some gasoline shipped in May, 1918, had given trouble in airplane engines in France, and that the loss of a number of planes had been attributed to its use. In a letter dated August 1, 1918, Col. U. G. Lyons, of the Fuel and Forage Division of the War Department, requested the Bureau of Mines to furnish inspectors for testing all shipments of gasoline.

Details were arranged at a conference between Col. Lyons and H. H. Hill in New York, August 26, 1918. C. R. Bopp was assigned to the task and spent practically his entire time from August 25 to December 1, 1918, inspecting shipments of gasoline for service overseas, from the Atlantic Refining Co., Philadelphia; Gulf Refining Co., Philadelphia; and the Standard Oil Co. of New Jersey; and the Tidewater Oil Co., Bayonne, N. J. Mr. Bopp inspected shipments representing 8,000,000 gallons of motor gasoline, 2,600,000 gallons of export aviation gasoline, and 725,000 gallons of fighting gasoline. W. G. Hiatt inspected at Gulf ports 55,000 barrels of motor gasoline for the American Expeditionary Forces.

EFFICIENCY IN RECOVERY OF GASOLINE FROM NATURAL GAS.

The attention of oil men was called to the feasibility of treating natural gas of any gasoline content, however small, at any pressure by the absorption process. As tests of the residual gases from compression plants showed incomplete recovery of gasoline vapors, the bureau pointed out that the gasoline in the residual gas can be saved at small cost by an auxiliary absorption unit.

Through the bureau's efforts companies recovering gasoline from natural gas by compression have come to realize the waste of gasoline in the gases discharged from water-cooled coils, and have put in compression or absorption units to save this gasoline. The average loss in the gas being discharged from the plants examined was about 0.3 gallon per 1,000 cubic feet. In one plant the adoption of recommendations by bureau engineers effected a saving equivalent to $120 a day.

The investigations of absorption and compression plants were conducted by W. P. Dykema, assistant petroleum engineer, aided by R. O. Neal, assistant chemical engineer.

COLLECTION AND DISSEMINATION OF DATA ON THE REFINERY INDUSTRY.

In the fall of 1917 maps showing the oil fields of the United States, storage centers, refinery capacity, and consumption statistics were prepared for the Fuel Administration by H. F. Mason, petroleum economist.

During the summer of 1917 a census of the refineries of the United States was made in order to determine the status of the refining industry, and to furnish the War and Navy Departments with reliable information as to the consumption of crude oil and the output of gasoline, kerosene, and other products. Practically every refinery was visited. The figures thus obtained were grouped according to districts in a report compiled at the Washington office. Messrs. Bowie, Tough, Beal, Wadsworth, Netzen, Hill, Jacobs, and Wiggins were engaged in this work.

After the United States Fuel Administration was established, the petroleum division of the Bureau of Mines, by request, continued to compile monthly statistics of the refining industry. The work was in charge of Mr. Mason.

On account of the value of petroleum coke in the electrode business and in the production of caustic soda and chlorine, a census of the manufacturers and the production of that commodity was made by Mr. Mason in December, 1917.

On account of the possible shortage of ammonia during the summer of 1918, estimates were prepared by Mr. Mason, in May, 1918, for the Fuel Administration, as to the amount of ammonia the oil refineries would need for the year. Mr. Mason made a similar investigation of the sulphuric acid used in the oil-refining industry for the first seven months of 1918. Also, he prepared other special reports for the Fuel Administration.

In September, 1917, a report on the supply of fuel oil available for naval purposes was prepared for the Shipping Board by J. H. Wiggins. A report was prepared in cooperation with members of the United States Geological Survey for the Council of National Defense, giving a resumé of the condition of the petroleum industry during April and May, 1917, especially as regards the supply of and the demand for crude petroleum.

A map and report showing the gasoline storage facilities in the United States was prepared for the Signal Corps in August, 1918, by Mr. Wiggins. He also prepared a report for the War Department in February, 1918, giving comparative data on costs, labor, etc., for transporting gasoline by pipe line and tank car in France.

In March, 1918, a map showing the oil fields of the world and their production was prepared for the United States Fuel Administration by A. W. Ambrose.

In January, 1918, C. H. Beal and A. W. Ambrose, in cooperation with members of the Geological Survey, prepared for the Shipping Board a report dealing with the need for Mexican oil in this country.

ESTIMATING FUTURE PRODUCTION OF OIL AND GAS, AND APPLICATION TO WAR REVENUE TAXATION.

Three months were spent in cooperative work with the Bureau of Internal Revenue on methods for estimating depletion charges in determining excess profits and income taxes for oil and gas properties. Because of the difficulties involved, no satisfactory method had been worked out and no final settlement of taxes on oil and gas properties had been made for three years. The method adopted by the Bureau of Internal Revenue was based on the methods devised by C. H. Beal and J. O. Lewis, petroleum engineers of the Bureau of Mines. Mr. Lewis had charge of the compilation of curves and data for the Mid-Continent oil fields, and Mr. Beal for the fields of California. A. R. Elliott assisted in compiling curves for the report. The principles adopted should benefit the petroleum industry and enable the Treasury Department to compute depletion allowances by an easy and equitable method.

During 1918, reports were prepared from time to time for the Capital Issues Committee on the issuance of stock by oil companies. For many of the properties a detailed field investigation and a thorough search of all available literature were required. Messrs. Dykema, Beal, and Ambrose conducted this work.

ESTIMATE OF COST OF GAS FOR NAVAL ORDNANCE PLANT.

The Navy Department requested the United States Geological Survey to prepare a report on the amount of gas available for the armor-plate plant to be constructed at Charleston, W. Va., and asked the Bureau of Mines to estimate the cost of producing the gas and of piping. The request to the Bureau of Mines was made about November 10, 1917, by Rear Admiral Ralph Earle, chief of the Bureau of Ordnance of the Navy.

A detailed estimate was made of the cost of drilling, of laying pipe, and of installing a compression plant for gas from the properties described by the Geological Survey, the cost of the gas being based on the survey's estimate of the volume. The report was made December 18, 1918, by Messrs. Wiggins and Dykema.

In September and October, 1918, at the request of the Secretary of the Navy, through Rear Admiral Earle, a report was made by R. V. Mills, petroleum technologist, and A. R. Elliott, assistant engineer, on the gas property from which it was proposed to furnish gas to the naval ordnance plant near Charleston.

VALUATION OF OIL PROPERTIES IN THE NAVAL PETROLEUM RESERVES.

In a letter dated January 18, 1918, Hon. Scott Ferris, chairman of the House Public Lands Committee, requested Secretary Lane to supply certain information on the petroleum industry. One of the most important and difficult questions asked was the value of the oil properties in the naval reserves. The information was compiled during February, 1918. The appraisal of the lands was made by Messrs. Beal and Ambrose.

EFFICIENCY IN DEVELOPMENT OF CALIFORNIA OIL FIELDS.

At the request of the Fuel Administration an investigation was made to determine the possibility of maintaining and of increasing the production of oil in California by drilling promiscuously, as in the past, and by drilling at selected locations, without encroaching upon the naval petroleum reserves. At that time the outlook was that the production of oil in California would have to increase to meet the war demand, and the only practicable way of doing this was to drill in selected localities where the maximum amount of oil could be obtained with the least labor and cost.

The work was done during April and May, 1918, by Messrs. Beal and Ambrose, with the assistance of R. E. Collom, chief deputy State oil and gas inspector of the California State Mining Bureau.

CENSUS OF OIL-WELL CASING.

The War Trade Board requested the Bureau of Mines to furnish information as to whether exports of oil-well casing should be restricted. A census of the amount of casing in reserve in the different oil fields of the country was begun about December 20, 1918, and lasted until February 1, 1919. Oil-pipe companies gave information as to past and probable future demand, past and estimated future production, and amount on hand of all kinds of casing. A careful study of the foreign oil fields was included as it was necessary to provide pipe for those fields that were supplying petroleum to the allied nations. Mr. Ambrose was in charge of this work.

ECONOMY IN USE OF OIL FUELS.

Various problems in connection with oil-fuel economies were investigated for the western division of the Fuel Administration. A number of fuel-saving devices were examined, such as burners, carburetor attachments, gages for small oil tanks in buildings, and gas radiators. Twenty-five boiler plants about San Francisco Bay and 226 power plants in California using fuel oil were inspected and recommendations made that effected a saving of 5 to 25 per cent at each plant. Demonstrations were given at San Francisco of the use of powdered coal as a substitute for oil fuel. This work was done by C. P. Bowie of the bureau, who acted as technical adviser of the western division, and by J. M. Wadsworth, petroleum engineer of the bureau, who served as State administrative engineer for the California Fuel Administration.

The Bureau of Mines, in cooperation with the National Automobile Chamber of Commerce and the Council of National Defense distributed copies of a poster showing how gasoline is wasted and giving instructions for preventing waste.

TESTS OF FUEL OIL MIXED WITH POWDERED COAL.

From May 1 to June 1, 1918, cooperative tests of colloidal fuel oil were conducted for the Submarine Defense Association of America. The fuel was composed of Navy fuel oil so mixed with powdered coal as to retain the coal in suspension. The tests were made on the U. S. S. *Gem* by Messrs. Wadsworth and Perdew.

PROSPECTING FOR OIL IN THE BRITISH ISLES.

In cooperation with the British Government, through its agents, Pearson & Son, the Bureau of Mines selected drillers and drilling material for prospecting for oil in the British Isles. The cooperative work was conducted chiefly by C. H. Beal, A. W. Ambrose, V. L. Conaghan, and M. A. La Velle, Messrs. Conaghan and La Velle, expert drillers of the Bureau of Mines, leaving the service of the bureau to take charge of the drilling work.

UTILIZATION OF OIL-SHALES.

In July, 1917, W. E. Perdew, in a trip through the oil-shale fields of Colorado and Utah, collected information on the companies organized, the proposed methods of retorting, and details of some special types of retorts. Samples of shale from various districts in this country were tested for their oil content. Mr. Perdew also compiled a résumé of the oil-shale industry. The work on oil shale was continued by M. J. Gavin and W. D. Bonner. Under supervision of D. T. Day, consulting chemist of the bureau, a retorting plant is in process of construction at Elko, Nev. The funds for erecting the plant have been provided by the Southern Pacific Co.

PROTECTION OF OIL AND GAS IN MID-CONTINENT FIELDS.

Work to prevent waste of oil and gas became of vital importance in war time. Much effort was spent on introducing methods for protecting oil and gas sands by mudding and cementing, and in assisting operators in difficult drilling jobs. Particularly valuable results were obtained in the Cushing field, Oklahoma, where the production of oil was increased several thousands of barrels daily by excluding water from the wells with cement on the recommendations of A. A. Hammer and B. H. Scott. In the Butler County field, Kansas, where water threatened to cause much damage, the bureau's engineers rendered timely assistance. Also, an educational campaign was conducted to arouse producers and call their attention to methods for preventing oil and gas sands from being flooded. The use of cement to shut off bottom water materially increased the production of many oil wells in Kansas and Oklahoma. This work was conducted by A. A. Hammer, B. H. Scott, H. R. Reuch, J. O. Lewis, F. B. Tough, and Thomas Curtin.

A similar investigation of the water problems in the Illinois fields was conducted in cooperation with the State geological survey during six months, April to September, 1918. In a well repaired according to the recommendations of the bureau engineers, the flow of water decreased to less than one-third and the settled production of oil more than doubled. As a common valuation of settled production in Illinois is $2,000 a barrel of daily output, the value of this property was increased by about $20,000.

The work was done by F. B. Tough, assisted by Frank Madden, of the Bureau of Mines, and by M. L. Nebel, assisted by S. W. Williston, of the State geological survey.

EXAMINATION OF THE SALT CREEK FIELD, WYOMING.

In September, 1918, the Secretary of the Interior requested the Bureau of Mines to examine the Salt Creek oil field of Wyoming. F. B. Tough and B. H. Scott, of the bureau, spent a month in that field and prepared a preliminary report, which showed that present methods of drilling and producing are wasteful. They suggested means of checking the waste and recommended that steps be taken to safeguard the equity that the Government has in the field.

MISCELLANEOUS LABORATORY TESTS AND ANALYSES.

Fuel oil used by the War Department at the Watertown Arsenal, Watervliet Arsenal, Fort Sill, and Panama is purchased on specifications. Samples of the consignments received are analyzed at the bureau's petroleum laboratory at Pittsburgh. During 1917 and 1918, samples representing purchases of more than $1,000,000 were analyzed.

At the request of the Ordnance Department of the Army, the Bureau of Mines inspected the wash oil used at certain toluene recovery plants. About 15 samples were examined, the work being done principally by N. L. Shoop, D. C. Dunn, and N. A. C. Smith.

While in London, Mr. Naramore was requested to obtain information concerning the available supply of American fuel oil that would meet British Admiralty specifications, which are more exacting than those of the American Navy. Tests conducted at the Pittsburgh petroleum laboratory showed that practically all distillates and some residuum fuel oils would meet the British viscosity requirement. The laboratory investigation was undertaken by J. P. Smootz and N. A. C. Smith.

Some experiments were conducted by H. H. Hill at the Pittsburgh petroleum laboratory for the American University station to ascertain the best oil for saturating Army blankets to be used as screens to exclude poison gases from dugouts.

During the summer of 1917, 25 samples of emulsified lubricating oil from the Washington Navy Yard were analyzed and tested for viscosity and water content by J. P. Smootz, N. L. Shoop, and N. A. C. Smith.

Various preparations alleged to increase the efficiency of gasoline were tested and found to be worthless. This work was done chiefly by Clarence Netzen, H. H. Hill, and C. R. Bopp.

Various products that were claimed to be satisfactory substitutes for motor gasoline were analyzed and engine tests of some of them were made. The analyses were made by E. W. Dean, H. H. Hill, and C. R. Bopp.

Among devices tested by the bureau which had been offered the Government for the duration of the war were the Swan process for reclaiming waste crank-case oils, the Armstrong-Godward vaporizer, and a so-called motor perfector, the last two being designed to increase the efficiency of gasoline.

Mr. Perdew investigated the Swan process for the bureau and made a favorable report. The bureau recommended the adoption of the process to the Council of National Defense, and later to the French Scientific Commission during its visit to Washington. The process was put into use by the French Government.

The Armstrong-Godward device was designed to vaporize the gasoline-air mixture delivered by a metering carbureter. After mechanical developments had been in progress about a year the device was pronounced ready for final tests at the Bureau of Standards. These tests had not been completed in November, 1918. The investigation was conducted by C. Naramore, E. W. Dean, W. E. Perdew, and E. F. Hewitt, consulting engineer.

Tests made by W. A. Jacobs in August, 1917, at Waukesha, Wis., of the "motor perfector," a device for adding moisture to the gasoline-air mixture in a carbureter, indicated that it was no better than others already on the market.

CRACKING OF OILS.

Experiments in cracking oil from the Humble field, Texas, showed that the distillates were suitable for the manufacture of gasoline by cracking processes. The experiments were made in May and June, 1918, by W. A. Jacobs and D. C. Dunn.

In May, 1918, through an arrangement with the Gasoline Corporation, which controls the Greenstreet cracking process, six engineers of the petroleum division witnessed a series of test runs with the process at a plant in East St. Louis, Ill.

RIFLE CORROSION PROBLEMS.

At the suggestion of Gen. Ainsworth, United States Army, retired, the petroleum division of the bureau studied the causes of corrosion in Army rifles after being cleaned. A number of preparations designed to prevent corrosion with a single cleaning were collected and analyzed. Results of firing tests, made in cooperation with the Pittsburgh division of the Ordnance Office, have been negative, as no signs of corrosion have appeared. The rifles were stored in a dry room after cleaning, and failure to develop aftercorrosion may have been due to this cause. The investigation is being continued. The tests were made by H. H. Hill, W. A. Jacobs, and N. A. C. Smith.

TESTS OF THE DUNN TRENCH HEATER.

Tests for the War Department of the Dunn oil heater showed that the heater had some advantages over the regulation Army stove for heating tents. C. R. Bopp supervised the tests, which were made at the War College Barracks, Washington, D. C., June 26 and 27, 1918.

PRODUCTION OF HELIUM FOR USE IN AIRSHIPS.

One of the great scientific and technical triumphs resulting from the application of exact knowledge and inventive genius to problems of military importance has been the large-scale production of helium, making this hitherto exceedingly rare gas available for use in balloons and dirigibles. The following account of the bureau's work in connection with helium was prepared by Dr. Andrew Stewart.

DISCOVERY OF HELIUM.

During the solar eclipse of August 18, 1868, P. J. C. Janssen noted in the solar chromosphere a bright yellow line near to but not identical with the D_1 and D_2 lines of sodium. Frankland and Lockyer called the line D_3 and, as it was referable to no known terrestrial substance, ascribed it to a hypothetical element which they called *helium*, from the Greek word *helios*, the sun.

In 1889, Dr. W. F. Hillebrand, of the United States Geological Survey, reported in the American Journal of Science that in analytical work on uraninite, which he began in 1888, he had obtained up to 2½ per cent by weight of a peculiar gas by treating the mineral with non-oxidizing inorganic acids. This gas had the characteristics of nitrogen, yet was different from any nitrogen known, and was thought by Hillebrand, on purely negative grounds, to be some allotropic form of that element. Lack of proper equipment, however, precluded any further researches on the subject. As a matter of fact, a certain percentage of the gas so found by Hillebrand was nitrogen.

The attention of Sir William Ramsay, the distinguished British chemist, was later drawn to Hillebrand's work, and he repeated the latter's experiments. He separated from cleveite, a variety of uraninite, a gas identical with that of Hillebrand, in the spectrum of which was found, with Sir William Crookes' aid, the D_3 line of Janssen, proving that in the gas there was present a new element, to which the name "helium," borrowed from Frankland and Lockyer, was given. On sparking this gas with oxygen to eliminate nitrogen, removing the excess oxygen with phosphorus, introducing the residual gas into an exhausted spectrum tube and passing an electric current through it, a bright yellow glow was obtained. Examined with a spectroscope, this glow gave rise to a series of lines in the red, green, and blue, but most prominent was the bright yellow line of

Janssen. In 1895 the characteristic line of this gas was discovered in the spectrum of the atmosphere by Kayser, and the element was isolated from the air by Ramsay and Travers in the same year.

OCCURRENCE AND PROPERTIES.

Helium is widely distributed in nature, but generally in minute quantities. The amount of it in the earth's atmosphere is exceedingly small, being present in the proportion of only 1 volume to 250,000 volumes of air. It has been found, as mentioned above, imprisoned within radioactive mineral substances, in the gases evolved from some thermal springs, in those emanating from volcanoes, and also in the natural gas of several gas fields in the United States (notably in Kansas, Oklahoma, Ohio, and also Texas). In some of the natural gas of this country helium is present in amounts as high as 2½ per cent. Of late, emissions of helium in comparatively large quantity have been reported from the boric acid soffione of Larderello, Tuscany, Italy. It issues from the earth at Stassfurt, Germany, and at Karlsbad. The natural gas in certain parts of Europe also contains helium, but in much smaller quantity than in our own.

Helium is one of a series of very rare, inert gases—other members being neon, argon, krypton and xenon—all of which are present in very small quantities in the atmosphere. Thus far every attempt to make these gases enter into chemical combination has failed. Helium is, next to hydrogen, the lightest of known substances. The gas, although twice as heavy as hydrogen, has in balloons a buoyancy or ascensional power of 92.6 per cent, as compared with the latter. The reason for this is that the buoyancy of a gas is measured not directly by its weight, but by the difference between its weight and that of the air displaced. Both hydrogen and helium are so light as compared with air that the difference in their lifting power is relatively insignificant. For example, 1,000 cubic feet of pure hydrogen will lift a weight of 75.14 pounds whereas the same amount of pure helium will lift 69.58 pounds, and a mixture, containing 85 per cent helium and 15 per cent hydrogen, which is the gas contemplated for all-around aircraft use, will have 93.4 per cent of the lifting power of hydrogen and will lift 70.18 pounds.

In its physical behavior, helium is the nearest approximation to the ideal perfect gas. It is monatomic and liquefies at even lower temperatures than hydrogen. By its evaporation in vacuum, Onnes, of Leyden, produced it in liquid form, first cooling it in solid hydrogen and then expanding it through a small nozzle. Liquefied helium is a colorless, mobile liquid, having a density of 0.122; hence it is the lightest liquid known. It boils at 4.5° absolute, and has a critical temperature of about 5° absolute, with a critical pressure of 2.75 atmospheres. By the rapid evaporation of liquid helium, a temperature below 2.5° absolute has been reached, but solid helium has not been produced. The low dielectric strength of helium permits electric discharges to pass through it with much greater ease than through most gases, and its conductivity for heat is very high.

Helium has been proved to be the end product of the emanations of radioactive substances, but the origin of its presence in natural gas has not been established. The helium content of some natural gas is relatively so large and the radiation from radioactive material is so small that derivation from such material would seem entirely out of the question. In the whole world not more than three ounces of radium have yet been isolated: but it must be remembered that radioactive material is very widely distributed in nature and that helium generation has been going on through countless ages of past time. A great many factors enter into consideration in this question, however, and it is difficult to set up an unassailable hypothesis regarding it.

Helium is of prime importance in aeronautics because of its great buoyancy, because of its rate of diffusion and consequent wastage through fabrics being only half that of hydrogen, and, above all, because of its chemical inertness. The effectiveness of hydrogen-filled dirigibles in war, owing to the high inflammability of hydrogen, was reduced to the vanishing point when means of combating them with incendiary propectiles were developed. Even under peace conditions, the great hydrogen-filled envelope of the dirigible constitutes a serious hazard because of possible ignition from atmospheric electricity, or from flames originating in the power plant of the craft. Helium, however, being absolutely inert, can not be ignited or exploded, and even mixtures containing certain amounts of hydrogen with helium can, as has been indicated above, be used with perfect safety in lighter-than-air craft.

INITIATION OF THE RESEARCH WORK.

Up to April, 1918, helium had been obtained only in extremely small amounts, as a curiosity in scientific laboratories, so that the total quantity separated in the whole world probably did not exceed 100 cubic feet, the cost of production being about $1,700 to $2,000 a cubic foot. Hence, although the possibility of helium as a lifting gas had been realized, its practical use seemed out of the question.

In the development of a process by which helium is now being produced in large quantities, the Bureau of Mines was the pioneer and has taken the leading part. In March, 1916, the Director of the

Bureau of Mines called the attention of Dr. F. G. Cottrell, chief metallurgist of the bureau, to a new process for air separation that embodied some novel and striking features, the bureau being interested then in the cheap production of oxygen for use in blast furnaces. This process had been evolved by Fred E. Norton, a graduate of the Massachusetts Institute of Technology and an engineer of wide experience and international reputation; Mr. Norton had pooled his patents with E. A. W. Jefferies and the consolidation became known as the Jefferies-Norton processes.

On February 28, 1915, Sir William Ramsay wrote to Dr. R. B. Moore, of the Bureau of Mines, who had worked with him in investigating the rare gases of the atmosphere. Sir William said, among other things: "I have investigated blowers—that is, coal-damp rush of gas—for helium for our Government. There does not appear to be anything in the English blowers, but I am getting samples from Canada and the States. The idea is to use helium for airships."

As the United States was not yet in the war and strict neutrality was being maintained by this country, these remarks were passed over by Dr. Moore, but he remembered that in 1907 Dr. H. P. Cady, of the University of Kansas, had found more than 1 per cent of helium in some natural gas from Kansas, and he realized what it would mean if such helium could be made available for balloons and dirigibles. Dr. Cady and D. F. McFarland were the first investigators to discover helium in natural gas; in 1907 they published a paper on the subject in the Journal of the American Chemical Society.[a]

In April, 1917, Dr. Moore attended a meeting of the American Chemical Society in Kansas City, Mo., where a paper on rare gases was read by Dr. C. W. Seibel, of the University of Kansas, who had worked under Dr. Cady. In the discussion that followed Dr. Moore pointed out the possibility of helium for war-balloon use if its separation from natural gas could be accomplished at a reasonable cost. He asked Dr. C. L. Parsons, chief chemist of the Bureau of Mines, who was present at the meeting, to take up the matter with bureau officials on his return to Washington, and this Dr. Parsons did.

In April, 1917, the Director of the Bureau of Mines placed G. A. Burrell (later Col. Burrell) in charge of the bureau's war-gas work. The Bureau of Mines had initiated this work, chiefly on gas masks, smoke screens, and toxic gases, in cooperation with the Army and the Navy. Mr. Burrell, who had been in charge of gas investigations for the Bureau of Mines from 1908 until 1916, had analyzed natural gas from different fields and knew of its helium content. He had thought of helium as a possible filler for balloons and dirigibles and had mentioned the matter to F. A. Lidbury, of the Oldbury Chemical Co., and from him learned of Sir William Ramsay's interest in the subject and of the presence of helium-bearing gas in Canada. Prof. Satterly and Prof. Patterson, of the University of Toronto, began experiments looking to the separation of helium from Canadian natural gas January 1, 1916. They used a Claude-system unit, at Hamilton, Ontario, but had much trouble in eliminating the heavier hydrocarbons of the natural gas.

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[a] Cady, H. P., and McFarland, D. F., the occurrence of helium in natural gas and the composition of natural gas: Jour. Am. Chem. Soc., vol. 29, 1907, pp. 1523 et seq.

Mr. Burrell was aware of the possibilities of the gases of the Petrolia field, in Texas, in regard to helium yield, and had samples sent to Dr. Cady for analysis. These samples contained about 1 per cent of the element. Thereupon, Mr. Burrell took up the question of utilizing this gas for helium recovery, with Dr. Cottrell and Dr. Moore, and the matter was discussed at length.

On May 12, 1917, Mr. Burrell wrote to Maj. C. DeF. Chandler (now Col. Chandler, of the Air Service), asking for an expression of opinion as to whether helium possessed sufficient advantages for use in balloons to warrant undertaking its production; and Messrs. Burrell and Moore, at the latter's suggestion, conferred with Maj. Chandler to ascertain whether the Army would consider furnishing the necessary funds for the project. G. O. Carter, of the Bureau of Steam Engineering of the Navy Department, learned from Maj. Chandler of this matter and became interested in it on behalf of the Navy Department.

At this time several conferences were held to discuss helium production from every angle, and opinions and views, all favoring it, were obtained from Dr. Cady, T. B. Ford, then in charge of the low-temperature laboratory of the Bureau of Standards, and O. P. Hood, chief mechanical engineer of the Bureau of Mines.

Dr. Cottrell advised Mr. Burrell to get in touch with Mr. Norton, for the possible employment of his process to separate helium from natural gas; accordingly, Mr. Norton was asked to come to Washington, where he arrived on June 4, 1917. Asked for a minimum estimate on the cost of a small experimental plant for the purpose, Mr. Norton estimated that a plant with a capacity of about 5,000 cubic feet of helium a day, to try out his process, might be constructed for about $28,000. With this data in hand, the Director of the Bureau of Mines did his utmost to push the investigation. On July 19, 1917, he addressed a letter to the Chief Signal Officer of the War Department, requesting an allotment of $28,000 to try out the Norton process, and on July 20, 1917, Secretary Lane wrote to the Secretary of War on the same subject.

Dr. Moore had suggested to Dr. Cottrell that the Linde Air Products Co. and the Air Reduction Co., established firms in the business of gas liquefaction and separation, be brought into the helium undertaking, and in this suggestion Dr. Cottrell had heartily concurred. So these firms and one other (the Lacy process) were asked to cooperate with the Government, and developments finally culminated in a recommendation on July 26, 1917, by the joint Army and Navy Airship Board, of an allotment of $100,000, in equal shares from Navy and Army appropriations, to make helium available as a substitute for hydrogen for balloons and dirigibles. It was further recommended that arrangements be made with the Bureau of Mines to carry out the undertaking. On July 31, 1917, the Aircraft Production Board recommended that this be done.

About this time Capt. R. B. Owens, of the Signal Corps, went to England on a special mission. He had become very enthusiastic as regards helium, especially the possibilities of the Norton process, and took with him a letter from Mr. Burrell, which he presented to the British Admiralty. This led to the dispatch to this country of Commander C. D. C. Bridge and Lieut. Commander S. R. Locock, as a commission on behalf of the Admiralty to canvass the helium situation in this country and to investigate the Norton process. Through Vice Admiral Richard H. Peirse the Admiralty had written to Dr. R. A. Millikan (later Lieut. Col. Millikan), of the National Research Council, on July 25, 1917, that, on the strength of researches instituted by the Board of Inventions and Research, it had decided to fit up an experimental plant to investigate whether the use of helium for airships was practicable. The Admiralty was anxious to obtain from the United States 100,000,000 cubic feet of the gas at once and to contract for a further supply of 1,000,000 cubic feet a week, being particularly interested in the Norton process on account of the possibility of its greatly reducing the cost of producing helium, which the Admiralty feared would be prohibitive by the older, established processes. It was estimated that the best that could be expected from these older processes would be about $80 a thousand cubic feet.

DEVELOPMENT OF PLANTS FOR HELIUM PRODUCTION.

After the $100,000, mentioned previously, was allotted for helium work, the director of the Bureau of Mines placed Mr. Burrell in charge of the investigation. Mr. Norton started the design of a plant along the lines of his process, he and Mr. P. McD. Biddison, chief engineer of the Ohio Fuel & Supply Co., being appointed consulting engineers of the bureau. With others they made a field survey to determine the best site for operations. Dr. Cady was appointed consulting chemist on the staff of the Bureau of Mines; he and his assistants at the University of Kansas, particularly Dr. C. W. Seibel, did a large amount of analytical and research experimental work on helium, the results of which influenced the designs of the various helium plants that were constructed later. Dr. Cady also made valuable researches into the limits of inflammability of helium and hydrogen mixtures, and carried out a number of experiments on the permeability of balloon fabrics by helium.

The work in connection with the helium undertaking began at this time to assume such proportions and to broaden so much that Mr. Burrell felt that he would be unable to carry it on properly, his time being fully occupied with other pressing duties. Therefore, Prof. W. H. Walker, of the Massachusetts Institute of Technology, was asked to assume charge. This he did for a few weeks, and possibly it was at his suggestion that, for purposes of secrecy as a war measure, the name "argon" was substituted for helium and the three experimental plants finally constructed became known as "argon" plants. After Prof. Walker gave up this charge to become chief of the Chemical Service Section of the War Department, with the rank of lieutenant colonel, the helium work was carried on mainly by Dr. F. G. Cottrell in Washington and Mr. Norton and Mr. Biddison in the field, all under Dr. Manning's personal supervision, with Mr. Carter acting for the Navy and the Army, and the Linde Air Products Co. and the Air Reduction Co. cooperating so far as their processes were concerned. Contracts were closed with these two companies in November, 1917, and a contract was made with the Lone Star Gas Co., controlling the Petrolia field, for a supply of gas. Construction of a Linde plant and a Claude (Air Reduction Co.) plant was begun at Fort Worth, Tex., almost immediately after the contracts were executed. These plants, known respectively as "Argon Plants 1 and 2," were completed in March and in May, 1918. On May 11, 1918, Profs. Satterly and Patterson visited Fort Worth to inspect the plants and expressed themselves as highly pleased with what they saw. On February 2, 1918, Dr. Manning assigned F. C. Czarnecki to Fort Worth as superintendent of the Linde and Claude plants; also, about the same time Mr. J. R. George, jr., was assigned to assist Dr. Cottrell in Washington in conducting the helium work, with special reference to the Norton process. On January 1, 1919, Dr. Andrew Stewart succeeded Mr. George. In the spring of 1919 Dr. L. H. Duschak was for some time associated with the helium project.

In June, 1918, the director of the Bureau of Mines placed Dr. R. B. Moore in general charge of all three helium plants, as his personal representative. After visiting New York for consultation with the presidents of the Linde and the Air Reduction Cos. and to arrange for even closer cooperation of those corporations with the Government, Dr. Moore went to Worcester, where he conferred with Mr. Norton, and then went directly to Fort Worth. He has been actively connected with the helium work up to date, June, 1919.

The coming to this country of the British commission, in the fall of 1917, stimulated the interest of the Army and the Navy in helium. At a conference called by the director of the Bureau of Mines in his office, those present, besides Dr. Manning, were Commander Bridge, Lieut. Commander Locock, of the British commission; Prof. John Satterly, of the University of Toronto; Lieut. Commander Arthur H. Marks, Bureau of Construction and Repair, United States Navy; Capt. P. Pleiss, of the Signal Corps, United States Army; G. O. Carter, Bureau of Steam Engineering, United States Navy; Dr. W. H. Walker, Dr. C. L. Parsons, G. A. Burrell, P. McD. Biddison, F. E. Norton, Dr. Marston T. Bogert, National Research Council; W. W. Birge, president of the Air Reduction Co.; and Dr. B. S. Lacy. The helium project was discussed exhaustively and, as a result, recommendation was made that an appropriation of $500,000 be made available from Army and Navy funds, in equal shares, for four separate experimental plants, under the systems of Linde, Claude, Norton, and Lacy, respectively; that is to say, all systems recognized as practicable were to be given a trial. Later, the Lacy system was dropped. The recommendation was transmitted to the proper authorities of the Army and the Navy.

The $500,000 allotment was appropriated pursuant to a resolution of the Aircraft Production Board of October 17, 1917, but, on the recommendation of the Navy Department, the Norton process was excluded from the benefits of any share thereof. This unexpected action was embarassing both to the Bureau of Mines and to the British commission, but the Navy Department adhered to it and induced the War Department to take the same stand. Not until, at the Navy Department's insistence, the Norton process had been submitted to the searching examination of the National Research Council and that body had, by the unanimous action of a special committee, reported favorably on it January 14, 1918, was a further sum of $100,000 made available for the Norton process, on January 18, 1918. The special committee of the National Research Council was composed of Dr. Harvey N. Davis, of Harvard University; Dr. Edgar Buckingham and Dr. C. W. Waidner, of the Bureau of Standards; Dr. W. S. Landis, of the Air Nitrates Corporation; and S. L. G. Knox, consulting mechanical engineer, and later scientific attaché of the United States Embassy at Rome.

The Norton process is the latest practicable development in liquefying and separating gases. The Linde process (Plant No. 1) depends upon the so-called Joule-Thomson effect, obtained by the sudden expansion of a highly compressed gas through a small orifice, or nozzle, and the consequent cooling of the gas, the process being elaborated into a self-intensive or cumulative cycle of heat interchange by causing the cooled gas, on escaping, to circulate around the tube leading the initial gas into the apparatus.

George Claude, of Paris, conceived the idea of a liquefaction cycle with an expansion engine interpolated. Although the Joule-Thomson effect is used in the Claude cycle (Plant No. 2), its value is reduced to a minimum because the compression of the gas in this system is lowered. The maximum cooling effect is produced by the expansion engine because the compressed gas, on expanding in the engine cylinder, is made to do work and thus its temperature is lowered.

In the Norton process (Plant No. 3) three expansion engines are used, liquid is throttled, and the heat interchanger and fractionating still are of new design. In the Linde system an enormous expenditure of power is demanded to compress the gas in order to obtain the maximum effect of throttling, and this energy is then wasted. The Claude system requires much less compression power, but in this system the energy stored in the compressed gas is also dissipated. In the Norton system the requirement for gas compression is reduced to a minimum by the interpolation of the multiple-expansion engines, and what is needed is conserved and reapplied through the energy developed by these engines. Thus the maximum cooling effect is obtained at a minimum cost.

Construction of the Bureau of Mines (Norton process) plant, later known as Argon Plant No. 3, on the enlarged basis of a maximum production of 30,000 cubic feet of helium a day, was begun on April 3, 1918, and was completed October 1, 1918. The cost was $148,398.29, the estimate for the plant enlarged to the proportions given having been $150,000. During the early construction work at this plant Mr. Biddison's services were of especial value; succeeding him, April 20, 1918, George A. Orrok was appointed consulting engineer of the Bureau of Mines and has since been identified in this capacity with Plant 3.

The first site picked for the helium plants was at Fort Worth, Tex., selected by Messrs. Norton and Biddison, but later the Otto, Kans., region was deemed more advisable, as the natural gas there is exceptionally rich in helium; the supply, however, proved inadequate. The Bureau of Mines plant (Norton process) was then located at Petrolia, Tex., adjacent, for obvious reasons, to the properties of the Lone Star Co., but the plants of the Linde Co. and the Air Reduction Co., because of their requiring more water and more power, were built at Fort Worth, about 100 miles to the southeast, and were supplied with Petrolia gas through the Lone Star Co.'s pipe line.

The approximate cost of Plant No. 1 (Linde) was $245,000; that of Plant No. 2 (Air Reduction) was $135,000. The capacity of the

the Navy and War Departments consented to the further operation of Plant No. 2, the Navy agreeing to furnish gas of a higher helium content from its new pipe line (contemplated under "Plan C") when this would be completed, about May, 1919.

The Bureau of Yards and Docks of the Navy Department took physical possession of Plant No. 2 and of the Government property of Plant No. 1, and assumed all expenses incurred by the Government incident to these plants as of April 1, 1919. Plant 1 has been dismantled and the Linde Air Products Co. is to repurchase such of its apparatus and equipment as it agreed to do under its contract with the Bureau of Mines. All helium produced at these plants has been placed in the custody of the Navy Department at its special request.

Although only $36,000 was provided for further operation of Plant No. 3, under "Plan C," a visit of inspection to this plant by Commanders H. N. Jenson and H. T. Dyer, Lieut. Commander Smith and G. O. Carter, on behalf of the Navy Department, about March 15, 1919, so convinced the Navy officials of the exceptional merits of the Norton process and the possibilities of the plant in respect to producing high-purity helium at a minimum cost that they became strongly in favor of affording it every opportunity to prove itself.

After overcoming many obstacles that could not be foreseen, Plant No. 3, on April 2, 1919, began to produce helium, and by 5.30 a. m. April 3, a purity of 19.8 per cent was reached. Helium production continued until that afternoon, when it became necessary to shut down for some minor repairs. On April 17, 21 per cent helium was made, and it is confidently expected that helium of the highest purity will soon be produced by this plant on a large scale.

On April 15, 1919, the Director of the Bureau of Mines called a conference in his office to discuss matters incident to the further operation of Plant 3 and kindred matters. There were present, besides the Director, Commander Jenson, representing the Navy; Col. C. DeF. Chandler, Col. A. L. Fuller, and Dr. H. N. Davis, representing the Army; and George A. Orrok, Fred E. Norton, L. H. Duschak, W. A. Ambrose, and Andrew Stewart, representatives of the Bureau of Mines. Fred E. Norton, who had come from Texas for the purpose, gave a detailed report on the operation of Plant No. 3 and described the need for alterations in and additions to its apparatus to insure the greatest possible efficiency. It was the sense of the meeting that liberal allotments of funds should be made for the continued experimental operation of the plant, with regard to its ultimately being run as a production unit. As a result of this conference, the Director of the Bureau of Mines recommended an allotment of $100,000 to make the alterations and acquire the equipment referred to. This fund has been made available, one-half by the Army and one-half by the Navy.

COOPERATING AGENCIES.

During the course of the work of the Bureau of Mines on processes of helium extraction, G. S. Rogers, of the United States Geological Survey, undertook a reconnoissance of the natural-gas fields in the United States with regard to their possible helium supply, so far as that might be judged by the analysis of samples of gas from existing wells and the study of geological conditions. In this work Dr. Cady and Dr. C. W. Seibel rendered valuable assistance in analytical investigations.

On April 3, 1918, the petroleum division of the Bureau of Mines began work on the possible discovery of natural gas yielding helium, and from April 3, 1918, to January 1, 1919 conducted intermittently administrative work relating to the conservation of such natural gas. Examinations as to the possibility of obtaining helium in Montana, North Dakota, Washington, California, and southern Canada were made by C. H. Beal, J. O. Lewis, and A. W. Ambrose.

Following a conference between Dr. Cottrell and members of the staff of the Bureau of Standards, the Director of the Bureau of Mines, on December 14, 1917, requested the Bureau of Standards to undertake the determination of certain physical properties of methane, especially with regard to its latent and specific heats and specific volumes over a wide range of pressures and temperatures. The purpose of this inquiry was to facilitate the experimental operation of the helium plants, in which liquid methane plays an important part.

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DEPARTMENT OF THE INTERIOR
ALBERT B. FALL, SECRETARY
BUREAU OF MINES
H. FOSTER BAIN, DIRECTOR

TREATMENT OF THE TUNGSTEN ORES OF BOULDER COUNTY, COLO.

BY

J. P. BONARDI and J. C. WILLIAMS

This Report is the Result of Work Done Under Cooperative Agreement With the Colorado School of Mines

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TREATMENT OF THE TUNGSTEN ORES OF BOULDER COUNTY, COLO.

By J. P. Bonardi and J. C. Williams.

INTRODUCTION.

Until the end of 1918 Boulder County ranked as one of the foremost tungsten-producing districts of the world, but in August, 1919, there was practically no production in the district. The operators there could not compete with tungsten ores from foreign sources, particularly China, Burma, and South America. As a result there has been a strong movement for placing a protective tariff on all imported tungsten ores and products; and it seems to be the general belief of tungsten producers that some form of protection is necessary to the maintenance of the industry in this country.

The stimulus, due to the great demand for tungsten alloys in the manufacture of munitions during the war, was felt in all the productive localities of the world. The ores of Boulder County were especially in demand on account of their comparative freedom from harmful impurities, and this demand greatly extended mining operations in the district and increased the efficiency of the general milling practice there.

SCOPE OF REPORT.

This paper deals largely with the development of milling practice in Boulder County, the ore-dressing methods in use, and the local manufacture of ferrotungsten and tungstic oxide. The geology of the district and the mining methods are treated only briefly. Local methods of analyzing ores, concentrates, ferrotungsten, and tungsten metal powder are discussed.

Figures giving the production of tungsten concentrates in the Boulder district, in the United States, and in the world, data on the buying of tungsten ores, costs of production, and the normal tungsten requirements in this country are presented. Imports and exports of tungsten ores of the United States are mentioned briefly.

The writers hope that this report will prove of interest to all those interested in the treatment of tungsten ores and the manufacture of tungsten acid, tungsten powder, and ferrotungsten.

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Acknowledgment is also due to J. G. Hibbs, F. C. Armstrong, Hugh F. Watts, W. R. Lewis, and J. G. Clark, of Boulder, for valuable information and helpful suggestions; as well as to R. B. Moore, now chief chemist of the United States Bureau of Mines, and Dr. V. C. Alderson and Robert M. Keeney, of the Colorado School of Mines.

GEOGRAPHICAL DISTRIBUTION OF ORE.

EXTENT OF DISTRICT.

Boulder County is one of the smallest counties in Colorado. It is rectangular in shape, measuring 32 miles east and west by 24 miles north and south. Its southeast corner is about 12 miles north of Denver.

As the accompanying map (fig. 1) shows, the tungsten district of Boulder County is small and rather well defined, having a length of about 12 miles in a northeast and southwest direction and a width of about 7 miles. This area includes nearly all of the scattered deposits lying outside of an area of about 10 square miles in which are found the richest and largest deposits. There are a few deposits near by in the north end of Gilpin County, near Rollinsville.

As the productive area is small and the field has been thoroughly prospected under the stimulus afforded by the high prices of 1914–15, the chances of finding new deposits are slight.

The more important towns of the district are Nederland, the chief business center; Lakewood and Stevens Camp, the principal active mining centers; Cardinal, Sugar Loaf, Magnolia, and Rollinsville. The altitude of Nederland is 8,237 feet.

The district is reached by the Denver, Boulder & Western Railroad and by the Denver & Salt Lake Railroad (Moffat Road). Sugar Loaf and Cardinal are on the Denver, Boulder & Western; Rollinsville is on the Moffat Road. Cardinal is about 2 miles north of

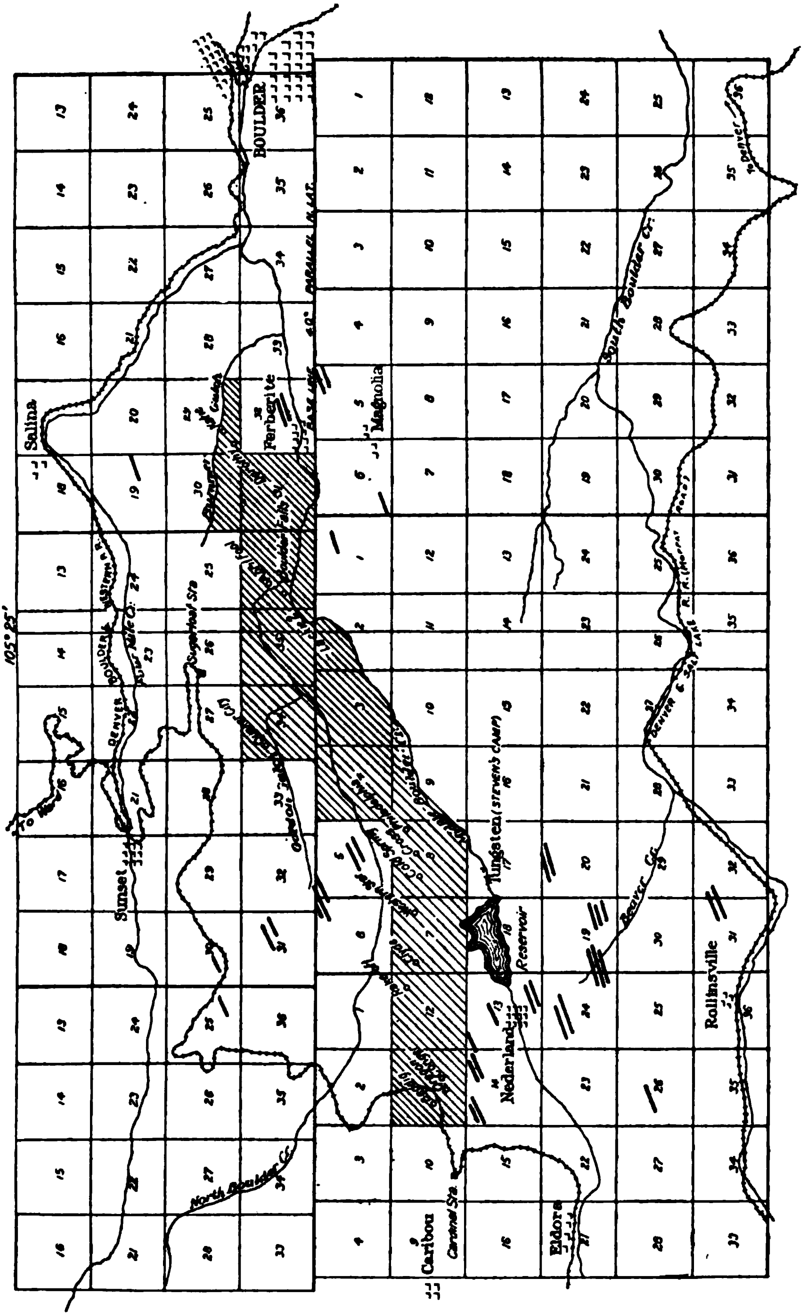

FIGURE 1.—The tungsten district of Boulder County, Colo. The tungsten district is shaded; mines outside this district are indicated by heavy diagonal lines.

Nederland, and Rollinsville is 4 miles south. Motor stage lines run from Boulder to the various towns of the district. An excellent highway, suitable for heavy motor truck hauling, runs up Middle Boulder Canyon from Boulder to Nederland, 17 miles, following the course of Middle Boulder Creek.

TUNGSTEN MILLS.

PRINCIPAL PRODUCING MINES OF THE DISTRICT.

At the time of the authors' visit to the district, January, 1919, the plants in operation, which are described in detail in this bulletin, were as follows: Black Metal Reduction Co., Boulder, tungstic oxide plant; Tungsten Products Co., Boulder, concentrator, ferroalloy and tungstic oxide plant; Grimm concentrating mill, Boulder Canyon, near Ferberite; Red Sign mill, Ferberite; Vasco mill, Tungsten (Stevens Camp); Wolf Tongue mill, Nederland; Rare Metals Ore Co. mill, Rollinsville.

Several other tungsten mills in the district were not in operation at the time. Among these were the following: Mojave-Boulder Mining Co., in Boulder Canyon, at the mouth of Bummer Gulch; Boulder Tungsten Production Co., Tungsten (Stevens Camp); Clarasdorf mill, Middle Boulder Creek; Black Swan mill, Salina; Primos Chemical plant, Copeland; Black Prince tungsten mill, Ferberite.

Mills that were not visited, though still in operation at the time the district was visited, were as follows: Tungsten Mining & Milling Co., Beaver Creek, near Rollinsville; Long Chance Mining & Milling Co., Beaver Creek, near Rollinsville; Primos Chemical Co., Lakewood.

Companies other than those mentioned above which have produced tungsten ores in the district are: Tungsten Mines Co., Tungsten Metals Co., Colorado Tungsten Co., Tungsten Mountain Mines Co., Duncan Mining Co., Commonwealth Tungsten Co., Gold & Silver Mining Co., April Fool Mining Co., Eagle Rock Co., Victor Mining Co.

In the Cold Spring district, near Lakewood, are the Cross mine, Orange Blossom, and Cold Spring, which are among the chief producers of the entire district. Near Nederland are the Clyde and other groups that have been large producers.

The ore treated by the Primos Chemical Co. is chiefly derived from the famous Conger mine, the Beddig, Lone Tree, Cork Screw, and Nos. 18 and 4 mines.

At Ferberite are the Red Sign mines which have supplied the Red Sign mill previously mentioned.

At Beaver Creek, near Rollinsville, are the following mines, which were large producers during 1918: Elsie, Long Chance, Mammoth, Manchester, Ebony, Fitzsimmons.

At Tungsten, Stevens Camp, are the 13 mines (numbered 1 to 13) of the Vasco Milling & Mining Co.

HISTORY OF TUNGSTEN MINING IN BOULDER COUNTY.

DEVELOPMENT PRIOR TO 1915.

The history of the tungsten industry in Boulder County illustrates well how a valuable mineral may be overlooked because of ignorance of its nature or because of its being of no economic value at the time of its discovery, and yet become of importance later when conditions change. In 1870 Sam P. Conger discovered the famous Caribou silver mine at Caribou, and a few months later the Boulder County mine, and Caribou was erroneously heralded as the richest silver district in Colorado and one of the largest in the world. The miners and prospectors who flocked into the district soon became familiar with a heavy dark mineral, found as "float," to which they gave various names, such as "heavy iron," "black iron," "hematite," and "barren silver." In the early part of the year 1900, Conger's partner, W. H. Wanamaker, who had seen the tungsten ores found in the Dragoon Mountains of Arizona, recognized this "float" as an ore of tungsten. Conger and Wanamaker kept their discovery a secret and in August, 1900, obtained a lease on the Boulder County ranch for the purpose of working the tungsten placers and developing veins. By the end of the year about 40 tons of high-grade ore had been taken out, which was purchased by the State Ore Sampling Co. of Denver. Half of it netted about $1.60 a unit. The other half was sold to Morris Jones, representing the Great Western Exploration & Reduction Co., at $60 per ton, or about $1 per unit. T. S. Waltemeyer, who was associated with Conger and Wanamaker, concentrated the ore.

In 1901 the reported production was 65 tons of concentrates having a tungstic acid content of 65 per cent. This was sold at about $2.25 a unit. In 1902 the market was extremely dull and much difficulty was experienced in selling the concentrates, but considerable prospecting and some development were done. The year 1903 was very favorable for tungsten producers, and in the Boulder County field there was considerable development. From 1903 until nearly the end of 1907 production was reasonably steady, but the financial depression late in 1907 arrested progress. The production in 1908 was about half of that in 1907; that of 1909 was only slightly below that of 1907; and the production in 1910 exceeded that of 1907.

Until the end of 1914 there was no development of particular importance.

DEVELOPMENT SINCE 1915.

By the middle of 1915 the demand for high-speed tool steel, of which tungsten is usually a component, became so great as to cause feverish activity among the owners of the known Boulder County deposits. Between January and June, 1916, there were 10 mills treating ferberite ores and six of these mills were new. By the end of June five old mills were being overhauled to supply some of the demand for milling capacity and three new ones were under construction.

In the tungsten mining camps the high prices brought about a boom similar to the booms caused by important gold discoveries. Nederland, which had been a little village of a few dozen homes, became almost overnight a bustling town of 3,000 or more inhabitants. Optimism prevailed throughout the entire district, and fabulous prices were asked for properties that showed nothing more than a streak of tungsten ore in the bottom of a 10-foot discovery shaft. Leases on old mine dumps and mill-tailing ponds were sought eagerly.

The price of tungsten rose to unheard-of heights, $75 a unit or more being the quotation for nearly two months, and at least one lot is known to have been sold for $105 a unit. Such high prices caused the large users to import tungsten ores from South America and later China, as these foreign ores could be laid down in this country very cheaply. As a result of these importations the market broke rapidly, and during the latter part of 1916, 1917, and nearly all of 1918 was fairly steady at about $25 a unit. The small demand, upon the signing of the armistice, together with the large importations and stocks held by dealers, forced the price down so much that early in 1919 all of the Boulder County producers were forced to suspend operations.

MINING AND MILLING.

MINING METHODS.

In general the ore bodies of the district are lenses a few feet long, but a few ore bodies have reached the exceptional length of 150 to 200 feet. These extend at right angles to the strike of the vein; they may reach a width of 14 feet. The veins range in thickness from a fraction of an inch to several inches. Most of the shafts are comparatively shallow—between 200 and 300 feet—although one, the Conger, is more than 1,000 feet deep. Generally the veins dip steeply and only rarely dip as low as 45°.

In development the usual practice is to crosscut from a vertical shaft until a vein is reached and then to follow it until it opens out. Then the vein is stoped by shrinkage stoping, the muck line being kept at a suitable distance from the back by drawing the excess ore through chutes into cars in the levels. If a high-grade streak is discovered, it is carefully broken out and kept separate from the lower-grade ore; usually a certain amount of hand sorting is done in the stopes. As a rule very little timbering is required.

A large amount of placer mining has been done, as the peculiar nature of the veins and of the ferberite is conducive to the formation of placer tungsten. Surface mining has been carried on extensively by open cuts, trenches, and shallow shafts. A large amount of rich ore has been recovered simply by picking up pieces of high-grade float and by using small jigs operated by hand or by a gasoline engine.

Expensive pumping equipment is not necessary, as the veins are comparatively dry.

Many of the mines are away from the creeks where most of the mills are situated, and the ore from these mines must be hauled by wagons or teams or, from the more easily accessible properties, by motor trucks.

MILLING METHODS.

As there are many small mines sending different kinds of ore to the mills of the large companies, the nature of the mill feed varies widely. The difficulties encountered in treating small lots of ores having different physical characteristics are thoroughly realized by all mill men. A mill well equipped for treating ores in which the ferberite occurs coarsely crystalline may not efficiently treat the "hornstone" ores, which require extremely fine grinding to liberate the tungsten mineral. For this reason a mill must be equipped to handle any ore that may be offered or take the chance of running short of ore or of making a poor recovery from the ore that is treated.

When the district was first visited by the writers (January, 1919) to get data for this report, the milling capacity largely exceeded the amount of ore offered for treatment, and the mills were running only 8 or 12 hours a day. By thus running on a comparatively light feed a good extraction can be readily obtained by a properly designed mill. The milling capacity of the district seems equal to any demands, even with the mines operating at full capacity.

POWER FOR MINING AND MILLING.

Steam or electric power is used at the larger properties. The smaller properties use electricity or, where only a small plant is required and a power line would be too expensive, gasoline engines. The electric power is furnished by the Colorado Power Co., which has a hydroelectric plant in Boulder Canyon, a few miles above Boulder, the reservoir being just east of Nederland, between that town and Stevens Camp.

The price of electric power on a monthly basis is 4 cents per kilowatt hour for the first 100 kilowatt hours, 1.3 cents for the next 150 kilowatt hours, and 0.2 cent for all above this amount. There is an additional monthly charge of 50 cents per horsepower demanded.

GEOLOGY.

George[1] describes the general geology of the district as follows:

> The area is wholly within the pre-Cambrian belt, called, in a broad way, the Front Range. The nearest sedimentary rocks, except recent stream and lake deposits, are 3 miles to the east. The most important rock is granite, generally more or less gneissoid. Next in importance is a granitic gneiss, frequently grading into quartz-mica-schist and mica-schist. While the areas occupied by these two units are, in a large way, well defined, there are numerous bodies of gneiss within the granite and numerous bodies of granite within the gneiss. In a number of places there is no well-defined contact line between the two, but a band or zone in which the two rocks are mingled and in which there is frequently a gradual transition from one type to the other.
>
> Cutting the gneissoid granite and the gneiss are granite intrusions in the form of dikes and irregular bodies. In the western and northern parts are many dikes, ranging in composition from acidic porphyries to latites, andesites, diabase, and basalt.

The tungsten-bearing veins are mostly in the pre-Cambrian granite, in gneisses that are believed to be of sedimentary origin and at the contact of the two. "Crossings" and "junctions" are likely to be accompanied by an enrichment. The miners believe that the schistose part of the gneiss is not favorable for the finding of ore and that the best ore is found in the acid rocks rather than in the basic.

ORE MINERALS.

FERBERITE.

Tungsten, in Boulder County ores, occurs almost entirely as ferberite, although some hübnerite is found, especially near Ward.. Scheelite is not of economic importance, but is occasionally found as small, almost microscopic fragments and is probably an alteration product from ferberite.

According to Hess,[2] ferberite "is that mineral of the wolframite series which is composed wholly, or almost wholly, of iron tungstate, and which, like other wolframites, crystallizes in the monoclinic system."

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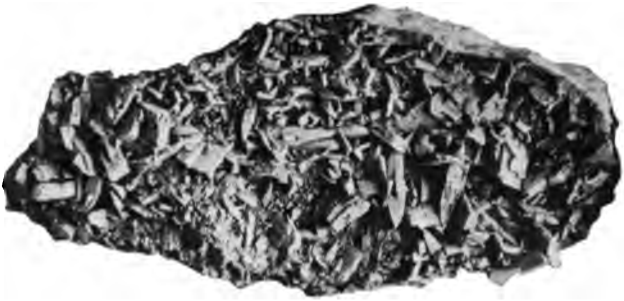
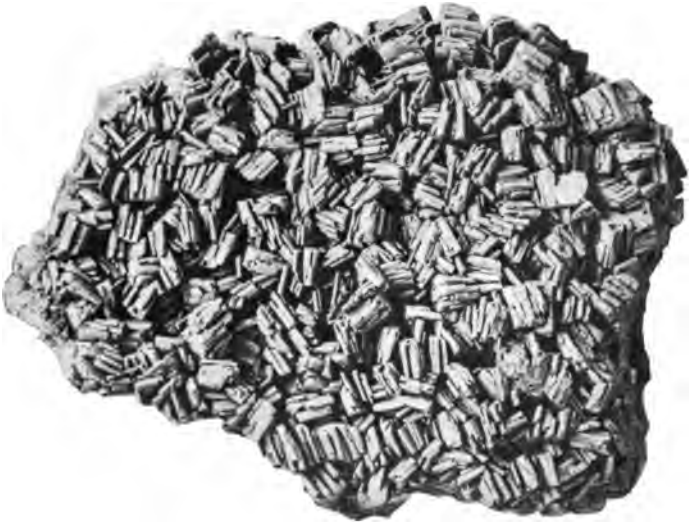
[1] George, R. D., The main tungsten area of Boulder County, Colo.: Colorado Geol. Surv., first report, 1908, pp. 7–103.

[2] Hess, Frank L., and Schaller, Waldemar T., Colorado ferberite and the wolframite series: U. S. Geol. Surv. Bull. 563, 1914, p. 7.

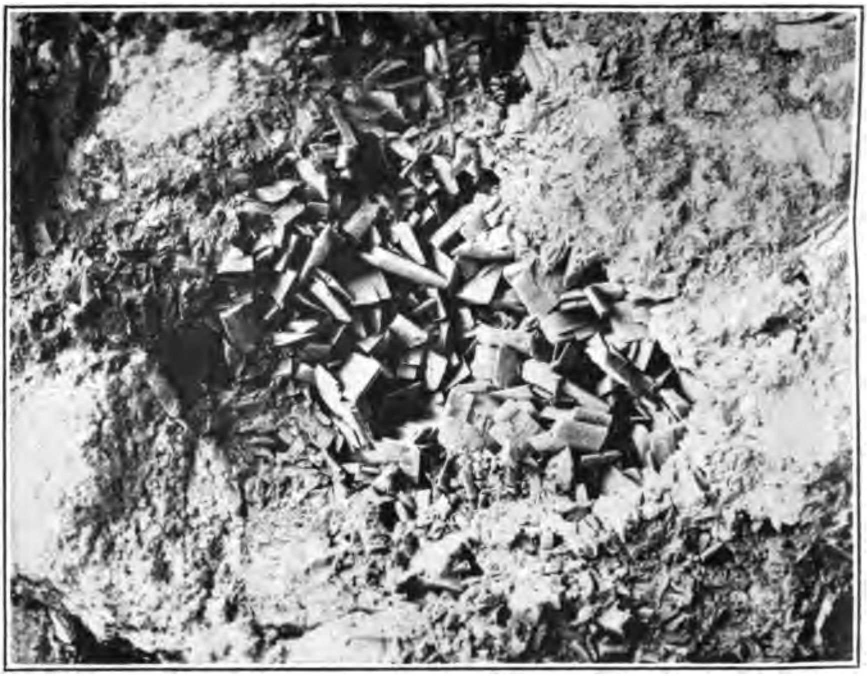


A. WEDGE-SHAPED FERBERITE CRYSTALS FROM "CROW PATENT," NEDERLAND, COLO. NATURAL SIZE.

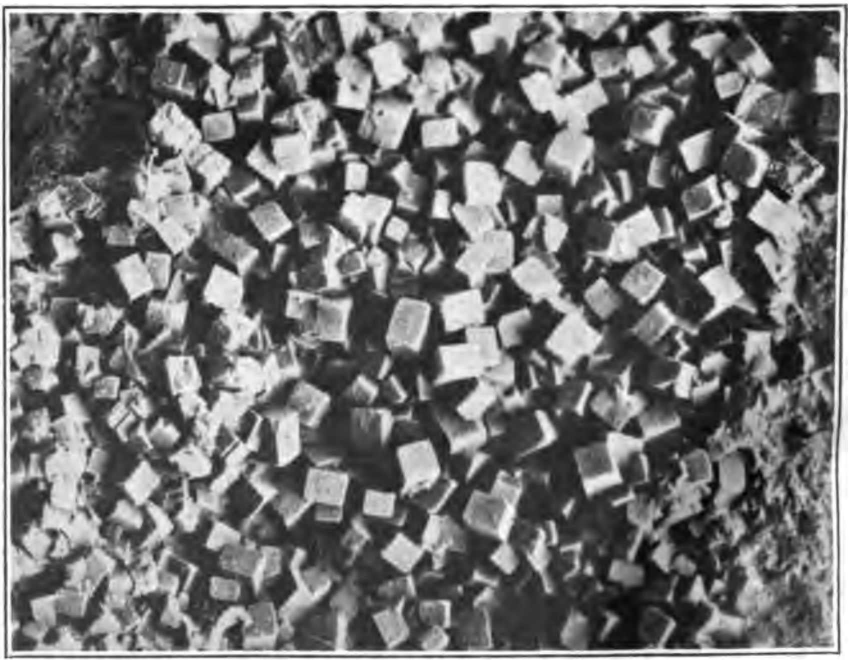


B. CRYSTALLIZED FERBERITE WITH ELONGATED, RHOMBIC CRYSTAL FACES, FROM NUGGET MINE, GRIFFIN COUNTY, COLO. NATURAL SIZE.

C. FERBERITE SPECIMEN, WEIGHT 2,100 POUNDS, ON EXHIBIT AT THE NATIONAL MUSEUM, WASHINGTON, D. C.



A. WEDGE-SHAPED FERBERITE CRYSTALS FROM HOOSIER MINE, NEDERLAND, COLO. ENLARGED TWO DIAMETERS.



B. CUBOID FERBERITE CRYSTALS FROM GEORGIA A. MINE, NEDERLAND, COLO. ENLARGED TEN DIAMETERS.

Except for its occurrence in the Boulder district, ferberite is comparatively rare, and in most places where it is found it occurs in small quantities only.

Ferberite, when pure, has a composition that may be expressed by the formula $FeWO_4$, corresponding to a content of tungstic oxide (WO_3) of about 76 per cent. Its specific gravity, determined on selected crystals, is 7.499, and its hardness about 5. Most of the ferberite in the Boulder district is jet black, but in a few places it is brown; the crystals and cleavage faces are usually lustrous black. The streak is characteristically chocolate-brown. Like other wolframites, ferberite crystallizes in the monoclinic system. The cleavage along *b* is perfect, and some specimens show a parting parallel to *a*.[3] The mineral is brittle with an uneven fracture, and is opaque, even in thin sections, as prepared for microscopic examination. A splinter of pure ferberite fuses to a crystalline nonmagnetic globule, but some of the ferberite from the Rogers tract is almost infusible before the blowpipe and is strongly magnetic after heating. The mineral occurs as wedge-shaped crystals, coarse to finely granular, and in the variety of ore locally known as "hornstone" the crystals are so intimately associated with quartz as to look like a stain; very commonly small crystals are coated with foreign material. Ferberite is very resistant to weathering; hence it has formed placer deposits that have been worked successfully. It is only very slightly soluble in acids. Some typical forms of ferberite are illustrated by Plates I and II.

QUALITATIVE TEST FOR TUNGSTEN.

Ferberite may sometimes be confused with magnetite, but differs from that mineral in color of streak and in being nonmagnetic toward a permanent magnet. The presence of tungsten may be confirmed as follows:

Grind the material to be tested very fine and boil with concentrated hydrochloric acid for about five minutes. Sulphuric acid may be used, but longer boiling is required. Nitric acid should not be used. After the material has settled, a fine yellow powder of tungstic acid is usually seen. Upon the addition of granulated zinc or tin a characteristic beautiful blue color is produced, which may be followed by violet and brown.

CHARACTERISTICS OF THE ORE.

Hess[4] describes the Boulder County ore as follows:

> The amount and character of the gangue in the veins differ greatly in different parts of the field. Quartz, the universal vein mineral, occurs in

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[3] Dana, E. S., System of Mineralogy, 6th ed., p. 983, 1909.

[4] Hess, Frank L., and Schaller, W. T., Colorado ferberite and the wolframite series: U. S. Geol. Survey Bull. 583, 1914, pp. 10–11.

smaller quantities in the ferberite veins than in most veins. Only a very little visibly crystallized quartz in the ferberite veins has come to my attention. There is, however, in most if not all of the veins of the northeastern part of the field, and in many of the wider veins of the southwestern part of the field, a very fine-grained gray or brown quartz, known as "bone," which has a fracture very much like that of chalcedony and which is largely a replacement of the country rock. The ferberite in the northeastern part of the field appears to be less well crystallized than in the southwestern half. It occurs in some of the veins in minute particles, mixed with quartz, so that in many places the ore is difficult to concentrate, although the total percentage of WO_3 may be as high or higher than in ores which are easily and profitably worked.

In the southwestern part of the field the narrower individual veins are made up almost wholly of ferberite, which has grown from both sides of the crevices and forms combs of small crystals that coalesce at their bases. The combs are of fairly uniform thickness, so that although in the narrower crevices they may have met and grown together, in the wider cracks they form bristling crusts of crystals on each side. In places these crystals are clean and bright, but generally very small, from one to three thirty-seconds of an inch (1 to 2 millimeters) across. Most of the crystals are covèred with a coating of impure chalcedony, which in some specimens is mixed with opal and in others contains much iron. The coating generally includes small crystals of ferberite, which are, of course, of a later generation than those on the walls. Some vugs 3 or 4 inches across are entirely filled with such a mixture.

In places the country rock, instead of being sheeted, is crushed into smaller, more nearly equidimensional fragments, which may not be over one-eighth inch (3 millimeters) in diameter. Ore in which fragments of rock no larger than half an inch (13 millimeters) across, are embedded in ferberite, is popularly known as "peanut" ore, from its resemblance to peanut candy. The fragments of the breccia vary greatly in size, and when the fragments are larger the breccia is characterized by vugs lined with ferberite crystals. Many of the veins have been opened several times, so that the ferberite itself is brecciated.

MINERALS ASSOCIATED WITH THE FERBERITE.

In the ferberite veins of the main part of the field there are few associated minerals except quartz. Near the gold-silver bearing areas there is, of course, a mixture of the minerals of the several vein groups. Hess and Schaller[5] have described in detail the minerals associated with ferberite taken from the Boulder district. These minerals comprise adularia, calcite, chalcedony, chalcopyrite, galena, gold and silver, hübnerite, wolframite, hamlinite, hematite (specular), limonite, magnetite, molybdenite, opal, pyrite, quartz, sheelite, sphalerite, and sylvanite. For a detailed account of the occurrence, association, and amounts of these minerals the reader is referred to the report cited.

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[5] Hess, F. L., and Schaller, W. T., Colorado ferberite and the wolframite series: U. S. Geol. Survey Bull. 583, 1914, pp. 12–17.

CONDITIONS GOVERNING MILLING PRACTICE.

Most of the mills in the district attempt to produce a concentrate containing 60 per cent tungstic oxide, although some that have refinery connections can market a product carrying as low as 40 per cent. The grade of concentrates produced may be considered according to the requirements of the three main uses to which they are put: (1) Ferrotungsten, (2) tungsten powder (metallic tungsten), (3) tungstic oxide (tungstic acid).

As noted above, the general standard is that the concentrates shall contain at least 60 per cent tungstic oxide. As pure ferberite contains 76 per cent tungstic oxide, the margin for obtaining the desired result by wet concentration may seem ample at first sight, especially as the specific gravity ferberite is about 7.5 and that of the gangue is between 2.5 and 3. But the ferberite is very brittle, slimes easily, and the great problems of milling ferberite ores are efficient handling of the slimes produced and crushing so as to liberate the ore particles with a minimum amount of sliming. An adjustment of the concentrating devices to produce a high-grade product results in a lower recovery, whereas attempts at a high recovery result in a comparatively low-grade product. This fact must be kept in mind when recoveries are considered; both the recovery and the grade of the product made should be known. A mill reporting a recovery of 75 per cent may seem to be doing poorer work than another reporting 90 per cent, but the first mill may be producing a concentrate of so much higher grade that comparatively it is doing excellent work.

By hand sorting some ores a small amount of 60 per cent ore may be obtained, but only rarely is a product better than 50 per cent made. Hand sorting usually does not pay, as the greater part of the ore remains to be crushed and concentrated.

The difference in amenability to treatment among ores from different localities, and even among ores from different parts of the same vein, adds to the difficulty of concentrating Boulder County ores. A low-grade ore may be more valuable than one of higher grade because it is easier to concentrate and yields a higher recovery. This difference depends entirely upon the physical association of the ferberite with the quartz. In some ores the ferberite crystals may be easily freed from the quartz; in treating such ores fine crushing is not necessary and sliming losses are greatly reduced. These ores grade imperceptibly into those locally known as "hornstone," so called from the blending of white and black, resembling that often seen near the tip of the horns of cattle. Such ores seem to be chalcedony with only enough ferberite to give a dark color. The appearance indicates a simultaneous deposition of the ferberite and quartz, with neither one impregnating the other.

This difference in mill-saving quality is strikingly brought out by the examination of results obtained by testing ores of different character. Several mill tests to illustrate this point are given by Victor G. Hills.[6] The tests were made in February, 1905, and the results are shown in Table 1.

The lots were tested on the same day, with the same tables, operated by the same men, and hence show unmistakably the difference in the nature of the ores:

TABLE 1.—*Mill tests made in 1905.*

Test number	1	2	3	4
Contents of crude ore, %WO_3	11.66	14.83	14.14	5.31
Contents of coarse concentrates, %WO_3	67.64	60.10	35.60	28.32
Contents of slime concentrates, %WO_3	66.60	64.40	44.20	49.52
Contents of total mixed concentrates, %WO_3	67.59	60.20	35.90	29.40
Per cent saved	83.40	72.60	52.70	32.20
Per cent of concentrates caught as slime	4.80	3.30	4.50	5.33
Per cent of WO_3 saved in the slime	4.65	3.53	5.50	9.11
Concentration ratio	6.95	5.60	4.82	17.18

Numbers 1 and 2 are from the west side of the district and numbers 3 and 4 are from the east side.

A selected piece from lot No. 3, without visible quartz or other impurity, apparently a solid mass of amorphous or cryptocrystalline ferberite, had a specific gravity of 3.01 and analyzed as follows:

Analysis of apparently amorphous ferberite.

	Per cent.
WO_3	30.96
Iron	10.20
Manganese	0.82
Silica	34.44

From an inspection of this analysis, in connection with columns 3 and 4 above, it is evident that a good grade of concentrate can be made from this ore only by extremely fine crushing, and then the slime loss must be abnormally high. Even then it is doubtful whether a 60 per cent product could be made.

MILLING PRACTICE.

CAPACITY OF TUNGSTEN MILLS.

The capacities of tungsten mills are not expressed by the number of tons treated in 24 hours but by the amount of tungstic oxide produced. The standard used in marketing tungsten is usually the "unit" 1 per cent of a ton, or 20 pounds. An ore carrying 5 per cent WO_3 is said to contain five units (100 pounds WO_3); and one carrying 60 per cent, 60 units (1,200 pounds WO_3). Obviously there is a great difference in two mill-runs of 25 tons each, when one contains

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[6] Hills, Victor G., Tungsten mining and milling: Proc. Colorado Sci. Soc., vol. 9, 1908–1911, p. 140.

5 per cent WO_3 and the other 20 per cent WO_3, although the tonnage is the same; the second run contains four times as many units treated as the first. The capacity of a mill in tons must state also the grade of ore.

EARLY PRACTICE.

In early development of this district the ores were concentrated in stamp mills designed for handling gold ores. Although the crushing equipment of these mills was practically unchanged, slime tables were added and the classifying and dewatering devices were modified, so that the table work was new. As was to be expected, such mills were not at all efficient in treating the ferberite ores. The average recovery prior to 1907 was probably about 50 per cent, and a recovery of 60 per cent was considered good work. However, this was not poor mill practice when the conditions under which the mills operated are considered. The market was uncertain and the prices paid for the concentrates were never very high. Many of the ore deposits were relatively small and the finances of many of the operating companies permitted only a minimum development. The use of existing mills was cheaper than the erection of new ones of better design as future conditions might not warrant the expenditure.

PRACTICE AT WOLF TONGUE MILL IN 1905.

A description of one of the old stamp mills, the Wolf Tongue mill at Nederland, is given by Van Wagenen.[7]

> The ore is dumped into bins from the wagons, thence over 2-inch grizzleys, through a 7 by 10 Blake crusher to a 20-stamp battery, the stamps of which weigh 1,000 pounds, drop 6 inches, 90 times a minute, through a 20-mesh long-slot screen; thence by launder to a hydraulic classifier which makes three products. The coarse goes to two No. 5 Wilfley tables, the middlings to a No. 3 Wilfley, and the slimes to two other No. 3 Wilfleys.
>
> Tables Nos. 1 and 2 (the No. 5 tables) make four products—a finished concentrate; a first middling, which is returned to the head of the table; a second middling, which goes to the Wilfley slimers; and a tailing. Tables Nos. 3 and 4 make two products—a finished concentrate and a tailing for the slimers. There are five 12-foot Wilfley slimers. The slimes from the five concentrating tables are brought together in a tank and distributed to four of the slimers, each of which makes three products—a finished concentrate, a finished tailing, and a middling taken from the four last panels. The middling goes to the fifth slimer where two products are made—a concentrate and a tailing. The mill treats 25 tons in 12 hours. The concentration is 15 to 1.

FIRST MILL ERECTED SOLELY FOR TREATING TUNGSTEN ORES.

In the summer of 1907 the Clarasdorf mill was erected on Middle Boulder Creek, about 11 miles from Boulder, on the Rogers tract, for the concentration of the ores produced from that property. This was the first mill to be erected solely for treating tungsten ores. A view of the Rogers tract and the Clarasdorf mill is shown by Plate III, *A* and *B*.

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[7] Van Wagenen, H. R., Tungsten in Colorado: Frenzel Prize Thesis, Colorado School of Mines, 1905.

An excellent description of the ore-dressing practice during 1906 and 1907 is given by Hills.[8]

> The mill of which the work is given below is that operated by the Colorado Tungsten Corporation. It is located in the city of Boulder, and was formerly known as the "Boyd Mill." There is a jaw crusher, ten 1,000-pound stamps, 4 standard Wilfley tables, 2 Wilfley slime tables, and 8 Monell slime tables, and stationary canvas. The table surface was large even when using all of the stamps; and the proportion of table area was sometimes increased by hanging up some of the stamps. Usually with ore reducing 10 or more to 1 all of the stamps were used, while with ore reducing 5 to 1, only five stamps were used with all of the tables, thus doubling the table area per ton of ore; and for other grades of ore stamps would be used in about the same proportion. With 10 stamps the capacity of the mill was a trifle more than 1 ton per hour. Experiments were made with screens of from 12 mesh to 40 mesh; different ores called for different sizes. Of the two veins producing the greater portion of the ore treated at this mill, 16 mesh proved the most efficient for one and 20 mesh for the other.
>
> The pulp was classified and passed over the four standard tables. The coarse tailing from the first table was discarded at once, carrying usually about 0.4 per cent WO_3. The tailing from the second table was reground. The tailings from the third and fourth tables went each to a separate Wilfley slime table. The tailings from these two tables, mixed, went to four Monell tables. The tailings from these went to a settling spitzkasten, from which the coarse material went to two more slime tables and the slime to the stationary canvas. There were many changes from time to time in the methods of distribution and classification, but the above is the plan in use during the greater part of the last year. Owing to the poor condition of the building and machinery there would usually be only six or seven of the eight slime tables in operation at any one time.
>
> During the last month of operation another large spitzkasten and two more slime tables were added, taking the finer material from the last classifier.
>
> The crude ore, the coarse concentrate, the slime concentrate, the coarse tailing, and the slime tailing were each sampled for every lot of ore. A mill record book was kept having 14 columns as follows: (A) Number of run; (B) number of shaft; (C) gross weight on cars; (D) percentage of moisture; (E) crude ore, dry weight; (F) assay of crude ore; (G) pounds of tungstic acid in ore; (H) pounds of concentrates saved; (I) assay of concentrates; (J) tungstic acid saved; (K) portion of tungstic acid in clean-up belonging to this lot; (L) total tungstic acid saved; (M) percentage saved; (N) ratio of concentration.
>
> There were also forms for daily and monthly reports.
>
> There is a record of 245 lots. Table 2 shows the best and the poorest of the eceonomic results. They are selected and arranged to illustrate three points: The difference in character of the ores from different veins, the difference in the saving qualities between the oxidized and the deep-level ore from the same vein, and the difference in the mill saving, with different appliances on the same kind of ore.

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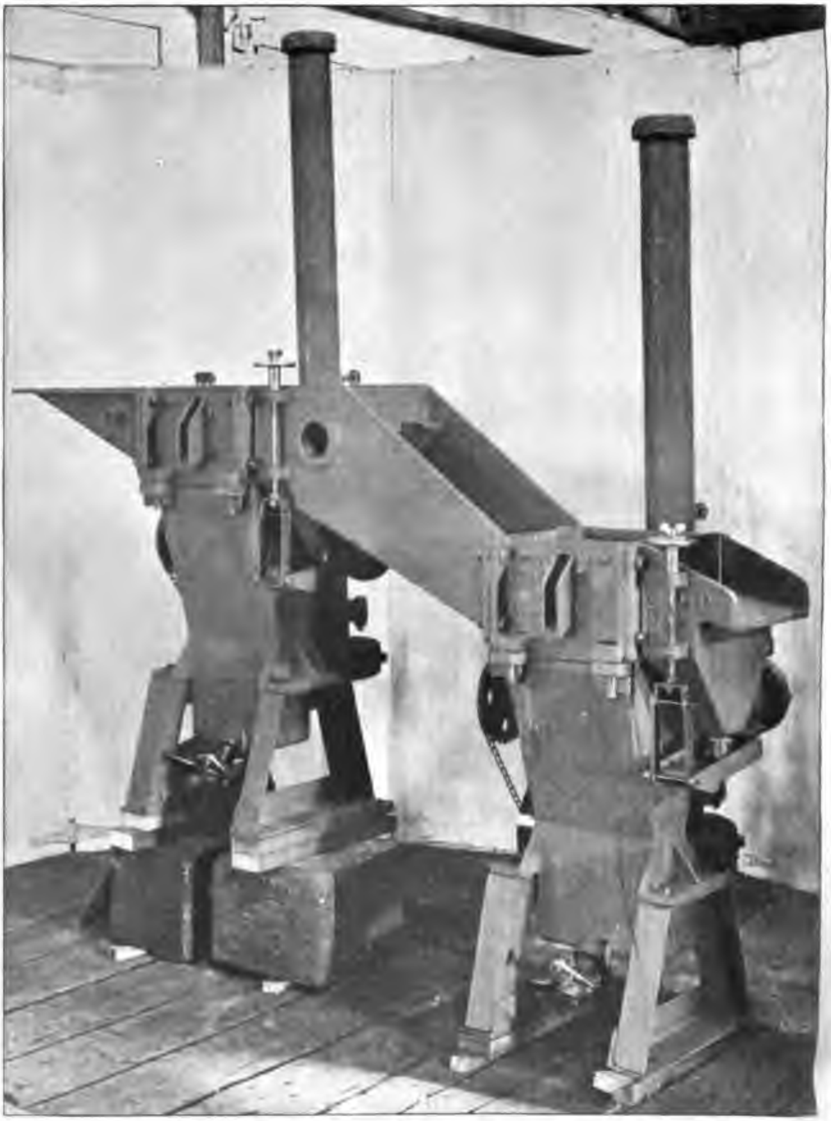
[8] Hills, Victor G., Tungsten mining and milling: Proc. Colorado Sci. Soc., vol. 9, 1908–1911, pp. 141–149.

A. ROGERS TRACT.

B. CLARASDORF MILL.

C. TRACT CONTAINING CONGER AND BEDDIG MINES.

D. MILL OF THE PRIMOS CHEMICAL CO.



TWO-COMPARTMENT RICHARDS PULSATING JIG.

TABLE 2.—*Representative mill work on tungsten ores during 1906 and 1907.*

Lot.	Mine.	Crude ore, per cent WO_3.	Concentration ratio.	Concentrates, per cent WO_3.	Saving, per cent.
1	A	16.57	4.70	65.41	85.22
2	B	12.43	7.96	54.49	56.19
3	C	6.07	181.81	40.37	5.03
4	D	8.07	8.69	62.17	88.91
5	D	12.03	7.10	65.12	76.44
6	D	6.09	12.05	62.11	85.62
7	D	6.48	11.91	64.88	84.78
8	D	6.82	11.49	62.15	80.35
9	D	6.97	11.15	62.45	81.35
10	D	5.83	13.61	62.25	80.00
11	D	7.50	11.04	62.93	79.06
12	D	4.31	18.58	59.13	74.84
13	D	5.16	14.57	61.42	82.74
14	D	5.54	13.28	61.20	87.38
15	E	9.51	8.93	62.59	75.11
16	E	9.94	8.27	60.48	75.08
17	E	8.67	9.43	61.38	76.50
18	E	10.08	8.53	60.48	73.47
19	E	8.11	9.68	60.41	77.46
20	E	5.31	14.72	60.27	81.58
21	E	5.27	14.15	61.01	86.52

Lots 1 to 7 are oxidized ores and all others are from deep levels.

Lot 1 is one of the best showings made from a small vein with little development.

Lot 2 is from a vein near Boulder Falls and is an example of one of the best ores ever treated from the east side of the district.

Lot 3 is the poorest ore ever brought to the mill (lowest in quantity that could be saved). It came from the east side of the district. It is readily seen that while the amount of tungstic acid in the crude ore compares favorably with the average ore of the district, the ore was commercially worthless. There was not enough of it saved to pay the cost of running it through the mill. It shows how an engineer, depending on his assays, without mill tests, might report favorably on a worthless mine. This, I understand, has actually been done in this district.

Lots 4, 5, 6, and 7 are of partly oxidized ores, and lots 8, 9, 10, 11, 12, and 13 are deep-level ores from the same vein, worked under substantially the same mill equipment. No. 14 is the only lot from this vein treated after the latest additions to the mill and is from the deepest level—180 feet on the dip or 140 feet vertically below the surface.

Lots 15, 16, 17, 18, and 19 are from another mine, 320 feet deep, at that time—1907—the deepest mine in the district. These lots were treated under the same mill arrangements as lots 8 to 13.

Lots 20 and 21 are the same ore as lots 15 to 19, treated with the latest additions to the mill.

The tungstic acid value of the several products, when working on normal deep-level ores, was usually as follows:

	Per cent.
Standard tables	62 -64
Wilfley slime tables	57 -59
Monell slime tables	35 -40
Stationary canvas	23 -27
Coarse tailings	.2- .6
Slime tailings	2 - 4

The slime tailing often varied widely with the ratio of concentration.

The output of the mill averaged above 60 per cent WO_3. Such few lots as carried below that figure were mixed with higher-grade lots so that nothing was marketed below the standard.

The lessons to be drawn from this exhibit are:

(*a*) That a comparison between the work of different mills is futile unless they were operating on the same kind of ore. Whenever any statement is made concerning the per cent of saving from any mill, three questions should at once be asked: First, from what part of the district was the ore produced? second, was it oxidized or unoxidized ore? third, what was the grade of the concentrate produced? I have heard that mills now operating in this district are marketing material down to a 30 per cent grade. It needs no demonstration to show that if this mill had been allowed to make any such low-grade product, the per cent of saving could have been readily increased.

(*b*) Oxidized ore concentrates a little better than the unaltered ore from the same vein. I regard this as due entirely to the difference in the physical condition of the gangue and the accompanying granite.

(*c*) The addition of the large settling and sizing spitzkasten, with two more tables and additional canvas surface, made a material increase in the saving, and, as would be expected, the additional saving was the greater with the ore which passed the finer mesh screen.

(*d*) The product of stationary canvas tables is low grade. It must be retreated on tables to secure a high-grade concentrate.

Table 3 gives some analyses of concentrates made at the Clarasdorf mill during 1907.

TABLE 3.—*Analyses of tungsten concentrates from Clarasdorf mill during 1907.*

Sample No	1	2	3	4	5	6	7	8	9
Tungsten oxide	67.99	59.62	59.52	58.21	56.84	48.44	38.46	38.59	52.38
Iron oxide	32.57	29.55	30.01	29.13	26.59	37.07	39.31	34.01	26.40
Iron suphide	....	2.57	2.09	2.34	2.55	.02	.51	2.59	3.25
Sulphur	....	1.37	1.11	1.24	1.36	.01	.27	1.38	1.73
Manganese oxide	1.02	.75	.76	.77	.75	2.65	3.10	2.67	.70
Phosphorus	Trace.	.01	.02	.01	.02	.01	.01	.01	.02
Silicon dioxide	8.32	8.42	7.54	9.56	11.25	11.51	18.61	22.13	17.25

These are mostly from the poorer grades of concentrate.

No. 1 is from the mine designated as mine "D" in the table of millwork. This vein produced, I believe, a better quality of ore than any that has so far been worked in the district.

Nos. 2, 3, 4, and 5 are from mine "E." They are of mixed concentrates and represent ores from the deepest workings.

No. 6 is from mine "B," near Boulder Falls. It was oxidized ore.

Nos. 7 and 8 are from veins at the east side of the district.

No. 9 is an analysis of a slime concentrate from one of the better veins near Nederland.

The following description of the Wolf Tongue mill, as it was in 1913, compared with Van Wagenen's description of it in 1905, shows the development of tungsten milling. Blake crusher and rolls have been substituted for stamps as primary crushers, jigs have been added, and the table work has been much improved.

PRACTICE AT WOLF TONGUE MILL IN 1913.[9]

The Wolf Tongue mill and power plant are adjacent to each other on Middle Boulder Creek, at Nederland. The ore is hauled from the various mines by wagons and dumped in four 15-ton ore pockets. Ore running better than 50 per cent WO_3 is shoveled to a 5 by 8 inch Sampson crusher set to 1 inch, which dumps to a chain-and-sprocket elevator with 4 by 4 by 6 inch cups and a 19-foot lift. This elevator, running at a speed of 88 feet a minute, dumps to a Vezin sampler, which makes a one-tenth cut, the sample reject going to a 4-ton bin, from which it is sacked and shipped.

The lower grade is shoveled to an 8 by 10 inch Colorado Iron Works Blake crusher, set to 1 inch and making 203 revolutions per minute. From the crusher it goes to a set of 14 by 32 inch MacFarlane cornish rolls, set to one-fourth inch and driven at a speed of 110 revolutions per minute by a 48-inch wooden "bull wheel" and friction. The rolls discharge to an elevator similar to the one described; this raises the ore 20 feet to a Vezin sampler, which makes a one-sixteenth cut, the reject going to an elevator which raises it 30 feet and chutes it to one of the three 30-ton bins, in which it is held until settlement is made with the lessee. These bins are provided in the mill with gates from which the ore may be chuted to the elevator which raised it to the bin and delivered through another chute to the jig bin, and on the outside with gates through which it may be unloaded and hauled away, if for any reason the lessee decides not to have it treated. All of the above machinery is driven by a 50-horsepower motor.

After settlement is made the ore is delivered to the elevator and chuted to a 20-ton bin, from which a plunger feeds it to the screen line, consisting of two 3 by 7 feet trommels with 6 and 12 meshes per inch. Each trommel sends its oversize to a two-compartment Harz jig. These jigs work through side and hutch, making concentrates in both compartments, that from the first compartment assaying 53 to 55 per cent and that from the second 25 per cent. They are sacked and shipped as separate products. A 15-horsepower motor drives the jigs and trommels.

The jig tailing goes to a 50-ton battery bin, from which it is fed by Challenge feeders to three of four five-stamp batteries. The stamps weigh 900 pounds each and drop 5 inches with even discharge, each stamp crushing 2 tons per 24 hours from ¼ inch through 18-mesh Ton-cap screens. The fourth battery is used for regrinding, as many stamps as may be necessary being dropped for this purpose. The even discharge is used to minimize the amount of slime produced. A 35-horsepower motor drives the batteries and two Frenier pumps mentioned later. The battery discharge goes to an 8 by 44 inch Frenier pump, which lifts the pulp 15 feet and discharges it to a tank, to which the undersize of the second trommel of the screen line flows by gravity.

On the table floor below the batteries are one Wilfley and five Card tables with speed and stroke as follows: 240 revolutions per minute, ¾ inch; 260 revolutions per minute, ½ inch; 270 revolutions per minute, ¼ inch; 270 revolutions per minute, ¼ inch; 285 revolutions per minute, ¼ inch; 258 revolutions per minute, ⅛ inch. The last table is a Wilfley, and is used for treating the recrushed product. Classification is by means of three small shaking screens designed by William Loach, manager of the Wolf Tongue, and peculiar to this plant. The screen consists of an eccentrically operated galvanized-iron hopper with a screen inside, so that the oversize is discharged to the table and the undersize passes out of the bottom of the hopper and on to the next screen.

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[9] Palmer, Leroy A., Tungsten in Boulder County, Colo.: Eng. and Min. Jour., vol. 96, July 19, 1913, p. 99.

The first three tables are provided with these sizers, as follows: 20-mesh, 240 ½-inch strokes; 40-mesh, 260 ¼-inch strokes; 90-mesh, 280 ¼-inch strokes. They are found to give satisfaction.

The first five tables, the Cards, make two grades of concentrate, a first grade running from 62 to 65 per cent, and a second grade running from 25 to 30 per cent. All tailing from the first table is reground, but the second and third make a rejection. The fourth and fifth treat the undersize of the last shaking screen and send all tailing to the slimers. All middling from the first three and last tables goes to an 8 by 44 inch Frenier pump, which elevates it 21 feet to the regrind battery, where it is passed through a 60-mesh screen and sent to the Wilfley table, which makes only second-grade concentrate and middling. A 20-horsepower motor drives the tables and sizers.

Slime is treated on three Monell slimers. This is a comparatively unknown machine, but has given good results on this ore. It consists of a rubber-edged canvas belt with an exposed surface of 5 by 18 feet and a transverse slope of 2 inches. Feed is discharged lengthwise for a length of 12 feet along the upper edge of the belt from a perforated pipe and wash water is fed on at the concentrate end. It has a belt speed of 1 foot per minute and receives 260 ½-inch strokes longitudinally from an eccentric without differential motion. The slimer feed consists of the middling of the fourth and fifth Card tables and the overflow of the table-feed tank, one slimer to each of the above products. The slimers make second-grade concentrate and the tailing is sent to a double-compartment concrete sump, 18 by 18 by 4 feet. While this tank is filling, quicklime, in the proportion of 20 pounds per 100 tons of tailings, is added to settle the colloids, which otherwise would remain in suspension almost indefinitely. As many of these fine particles consist of valuable mineral too fine to be recovered by any process yet devised, it is sometimes found advisable to ship the slimer tailing as second-grade concentrate. All tailing of ore running over 20 per cent WO_3 is treated in these tanks and either shipped or pumped back by an 8 by 44 inch Frenier pump and re-treated. A 15-horsepower motor drives the slimers, Frenier pump, 2-inch centrifugal pump for delivering slime concentrate to the drier, and a small compressor for cleaning the motors.

The concentrates are dried and shipped to the Firth-Sterling Steel Co., McKeesport, Pa., where their principal use is in the manufacture of ferrotungsten. First grade is settled for at the market quotation and second grade at about 5 per cent better than the same grade in ore.

Machines.	Number of motors.	Horsepower.
1 crusher, 1 set rolls, 2 elevators, 1 sampler	1	50
2 trommels, 2 jigs	1	15
2 pumps, 20 stamps	1	35
6 tables, 3 sizers	1	20
3 slimers, 2 pumps, 1 compressor	1	15
Total horsepower		135

Tailing is settled consecutively in three ponds before the overflow is discharged to the creek. The mill operates 12 hours per day with superintendent, foreman, sampler, jigman, tableman, weigher, and a mechanic who makes repairs in the mill and attends to the power house. The extraction is claimed as from 88 to 90 per cent. This is possible when the slime tailing is shipped and the only rejection made is from three of the Card tables.

GENERAL MILLING PRACTICE IN 1916.

In an article by Wolf and Barbour[10] the general milling practice of the district as carried on in 1916 is described as follows:

> The ore is passed over a 2-inch grizzly, from which the undersize is passed to a stamp battery. The oversize is crushed and delivered to the stamps. The stamps used vary in weight from 750 to 950 pounds each. The number of drops per minute ranges from 70 to 90. The drop is usually low and ranges from 4½ to 6 inches, and from 12 to 20 mesh screens are used. The pulp from the batteries is classified and distributed to Wilfley or Card tables, upon which the first concentrate is made. The tailings are frequently classified and again delivered to tables, or sometimes Frue vanners. Both 4 and 6 foot vanners are in use. The tailings from the second concentration on either tables or vanners are sometimes recrushed in a Huntington mill with a 60-mesh screen, and the product is treated on slime or canvas tables. This outline is subject to considerable modification in different mills. In some of the plants ordinary Harz jigs are employed to advantage and revolving screens are used for sizing. Both Wilfley and Card tables are in use, but the latter seem to be the favorite.

SUMMARY OF MILL PRACTICE.

Present milling practice for the tungsten ores of Boulder County is characterized by stage crushing and screening; each reduction is followed by concentration by means of the devices best adapted to each size. As a rule, the first concentration is done by jigs, either the Harz type or the Richards pulsating jig. The feed to these coarse jigs varies in size from that passing a 4-mesh screen to that which is retained by a 20-mesh screen. Probably the average size of the coarse concentrates is about 10 mesh. Jig tails are reground by rolls, and after screening may be treated in "fine" jigs. These are of the launder type of Richards jig and produce a concentrate which averages about 20 mesh in size. Where there are no "fine" jigs, the reground tails from the coarse jig are treated on "coarse" tables. Table tails are reground and prepared for the slime tables; table middlings may be reground and put into a middling circuit, or held for separate treatment when enough accumulates.

Crushing is carried to such an extent that when the tailings are discharged practically all of the mineral has been liberated. This means that a large proportion, which varies with the nature of the ore, will pass a 100-mesh screen. Screening out a portion of the sand table tailing low in tungsten and removing it from the mill circuit might pay, as this would lighten the load on the slime-treating devices, but the loss of tungsten might be overbalanced by the cost of saving it.

After the ore is crushed to three-fourths to 1 inch by means of jaw-crushers, rolls are used for preparation of the mill feed. Ball mills of various types are used for regrinding, and in one mill, the Wolf Tongue, stamps are used to regrind the jig tailing.

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[10] Wolf, Harry J., and Barbour, Percy P., The Boulder County tungsten district, Colo.: Eng. and Min. Jour., vol. 102, July 22, 1916, pp. 165–169.

Within the last few years stamps have been abandoned for fine crushing and jigs have been introduced to save the coarse mineral. In the stamp batteries coarse ferberite slimes readily because of its brittleness, and is subjected to undue crushing as its high specific gravity causes it to settle back on the mortar.

Canvas tables are installed in all of the mills visited with the exception of the Vasco, but the canvas plant in the mill of the Tungsten Products Co.. was not in use. The use of canvas tables is said to result in a recovery amounting to about 10 per cent of the total. However, they make a low-grade product, about 20 per cent WO_3, which requires cleaning on tables.

Extractions, as shown by company reports, vary from 75 to 90 per cent, according to the grade and character of the ore treated and the grade of the concentrate made. Possibly the best extraction under favorable conditions when a 60 per cent product is made is 90 per cent.

GRADE OF CONCENTRATES.

The concentrates are of two kinds—"crude" and "high-grade." The crude concentrates are utilized in the manufacture of ferrotungsten and tungstic oxide. These two products are basic for producing all compounds containing tungsten. All ferrotungsten is used as a means of introducing tungsten into ferroallows, whereas tungstic oxide is used to produce tungsten powder and chemicals. Tungsten powder is extensively used as a means of introducing tungsten into ferro and other alloys.

The high-grade concentrate—60 per cent WO_3 or more—is desirable for the manufacture of ferrotungsten. In making tungstic oxide by the hydrochloric-acid process the material used may range from 20 to 40 per cent WO_3, but in using the soda fusion method, a higher grade, 40 to 50 per cent, is desired, although at the plant of the Black Metal Reduction Co. ore mining as low as 12 per cent is used in a fusion process.

SAMPLING.

The usual sampling device is the Vezin sampler, set to cut from one-tenth to one-twentieth of the ore stream, or two Vezin samplers may be used, making a final cut of one-fortieth. The ore is crushed so that it passes a screen ranging from 4 to 8 mesh. The sample obtained by the Vezin sampler, if 300 pounds or less, is further reduced in size by sample crushers and rolls and is divided by hand by means of a sampler of the Jones-splitter type. If more than 300 pounds, the sample is again passed through the Vezin. After being sampled the ore, if it is a custom lot, is retained in a separate bin, called the settlement bin, until a satisfactory arrangement is made with the seller.

Another form of sampling device is known as the chain and cup, in which cups are supported between endless traveling chains. The quantity removed as a sample may be regulated by varying the number of cups. The cups passing under the ore stream retain a portion, while the rest of the stream drops between the chains. After the sample is taken it may be coned and quartered or passed through a splitter, and suitable reduction in size made by the sample crusher and rolls.

The final sample for assay is ground so that it passes 80 mesh and is about 1 pound in weight. This is split into the necessary number of assay pulps.

ASSAY OF ORES AND CONCENTRATES.

STANDARD METHOD.

The following method, developed by Hugh F. Watts, of Boulder, is used throughout the district as the standard for control and umpire work:

In preparing the sample for assay an amount slightly in excess of the weight desired is ground to a very fine powder by means of a mechanical agate mortar. Where a number of samples are to be prepared this machine is indispensable.

One gram of ore is weighed into a 250-c. c. beaker and treated with 60 c. c. HCl, or with ores carrying 5 per cent or less WO_3, 2 grams are weighed into a 150-c. c. beaker and treated with 40 c. c. HCl. Stir, cover with a watch glass, and digest for about two hours on a hot plate until the acid has evaporated to 10 or 15 c. c. Remove from hot plate and break up crust; add 40 c. c. of a mixture of one part HNO_3 to 10 parts HCl and digest again for one hour. Dilute with 50 c. c. H_2O, containing 1 c. c. HCl. Allow to settle and filter by decantation on a 12.5-cm. Swedish No. 2 or Whatman No. 30 filter paper. Wash three times with acid wash water (1 part HCl to 50 parts H_3O). To the residue in the beaker add 20 c. c. of a mixture containing 1,200 c. c. H_2O, 400 c. c. NH_4OH, and 20 c. c. HCl. Decant through filter into a No. 3 Coors porcelain crucible. Wash five times with a few cubic centimeters of the ammonia solution. Evaporate the filtrate to dryness, ignite, and weigh as WO_3. With low-grade ores it may be necessary to evaporate in a platinum crucible and treat the residue with hydrofluoric acid for the removal of silica.

In assaying Chinese ores, certain hübnerites, wolframites, and scheelites, after the ore has been treated as described above for ferberites, as much as 2 or 3 per cent of the total tungsten may remain with the insoluble matter. The residue is fused with the regular alkali carbonate mixture and leached with water; the iron and other residues are filtered off, and the filtrate, which contains the soluble tungstate, is titrated with nitric acid, added from a burette, until tungstate, is titrated with nitric acid, added from a burette, until to drive off CO_2 and the mercurous nitrate added, which precipitates tungsten as mercurous tungstate. As the ore had been previously treated with acids this precipitate will most likely be free from those interfering elements that are likely to follow tungsten, such as chromium, molybdenum, vanadium, tin, etc. After the addition of the mercurous nitrate the solution will still be slightly acid, and unless it is neutralized some tungsten will remain in solution. The solution is neutralized with Na_2CO_3 and heated to boiling; the precipitate is allowed to settle, filtered on a good grade of quantitative paper, and washed with water containing 2 per cent mercurous nitrate. The filter paper is then ignited in a platinum crucible and the WO_3 weighed. The contained silica can then be driven off by HF. Tin, if present, can be volatilized as stannic chloride by ignition with ammonium chloride. The mercurous nitrate may be prepared by digesting 2 or 3 ounces of mercury for about two hours with 25 c. c. of nitric acid (conc.) in 100 c. c. water, and allowing it to stand over night on a hot plate kept below the boiling point. This solution when diluted to about 400 c. c. will give a saturated solution with a minimum of free acid.

SPECIFIC GRAVITY METHODS FOR APPROXIMATE ANALYSIS OF ORES.

A specific gravity method for obtaining the approximate tungsten content of an ore is commonly used in the tungsten districts. The method used in the Boulder district is described by Hess as follows:[11]

> The Wolf Tongue Mining Co. originated a method which has been used by it and others on the ferberite ores of the Boulder field with excellent results, and the constants used there have been found serviceable in other fields. The mode of operation is as follows:
>
> The articles needed are a flask holding about 1,500 c. c. of water and scales weighing in grams up to 3 or 4 kilos. The flask is counterbalanced, then 1,500 grams of water is weighed into it and the height marked on the neck.
>
> For determinations, 1,300 grams of water is weighed into the flask and then dry ore is poured in until the water is raised to the 1,500 gram (c. c.) mark. This means, of course, that the ore occupies 200 c. c. and that an equal bulk of water weighs 200 grams.

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[11] Hess, F. L., Tungsten minerals and deposits: U. S. Geol. Survey Bull. 652, 1917, p. 71.

The weight of water in the flask, 1,300 grams, is subtracted from the total weight, and the difference, which is the weight of the ore, is divided by 200 grams, the weight of the water displaced, thus giving the specific gravity, which is compared with a table giving the equivalent percentage of WO_3.

At the Wolf Tongue mill the table has been elaborated so that weights may be directly read into percentages by referring to Table 4, as shown below.

The figures given in Table 4 are not exact specific gravities, but are close enough to give valuable data as to the probable metallic content of the ore. Such a method is applicable whenever the ore contains no other heavy minerals and the gangue is of fairly constant composition. Corrections would have to be made for other ores. For example, the specific gravity of the Boulder ferberite is 7.499, or, say, 7.5, and the specific gravity of the scheelite ore, if free from heavy minerals, such as galena, pyrite, and hematite, will be somewhat lower.

TABLE 4.—*Percentage of tungsten trioxide indicated by weights of 200 c. c. of ferberite ore plus 1,300 c. c. of water, in the Boulder field, Colo.*

Weight of ore and water (grams).	Specific gravity corresponding to weight of ore.	Specific gravity for given percentages of ore.	Percentage of ore.	Weight of ore and water (grams).	Specific gravity corresponding to weight of ore.	Specific gravity for given percentages of ore.	Percentage of ore.	Weight of ore and water (grams).	Specific gravity corresponding to weight of ore.	Specific gravity for given percentages of ore.	Percentage of ore.
1,815	2.575	....	....	2,050	3.725	3.717	32	2,280	4.900	....	....
1,820	2.600	....	....	2,055	3.750	3.761	33	2,285	4.925	4.933	53
1,825	2.625	2.650	....	2,060	3.775	....	....	2,290	4.950	....	....
1,830	2.650	2.650	....	2,065	3.800	3.805	34	2,295	4.975	....	....
1,835	2.675	....	....	2,070	3.825	....	....	2,300	5.000	5.008	54
1,840	2.700	....	....	2,075	3.850	3.850	35	2,305	5.025	....	....
1,845	2.725	2.724	1	2,080	3.900	3.899	36	2,310	5.050	....	....
1,850	2.750	2.748	2	2,085	3.925	....	....	2,315	5.075	5.088	55
1,855	2.775	2.772	3	2,090	3.950	3.948	37	2,320	5.100	....	....
1,860	2.800	2.796	4	2,095	3.975	....	....	2,325	5.125	....	....
1,865	2.825	2.821	5	2,100	4.000	3.997	38	2,330	5.150	....	....
1,870	2.850	2.847	6	2,105	4.025	....	....	2,335	5.175	5.177	56
1,875	2.875	2.873	7	2,110	4.050	4.046	39	2,340	5.200	....	....
1,880	2.900	2.899	8	2,115	4.075	....	....	2,345	5.225	....	....
1,885	2.925	2.925	9	2,120	4.100	4.094	40	2,350	5.250	....	....
1,890	2.950	2.951	10	2,125	4.125	....	....	2,355	5.275	5.266	57
1,895	2.975	2.979	11	2,130	4.150	4.155	41	2,360	5.300	....	....
1,900	3.000	3.007	12	2,135	4.175	....	....	2,365	5.325	....	....
1,905	3.025	3.035	13	2,140	4.200	4.211	42	2,370	5.350	5.355	58
1,910	3.050	3.063	14	2,145	4.225	....	....	2,375	5.375	....	....
1,915	3.075	....	....	2,150	4.250	....	....	2,380	5.400	....	....
1,920	3.100	3.096	15	2,155	4.275	4.267	43	2,385	5.425	....	....
1,925	3.125	3.129	16	2,160	4.300	....	....	2,390	5.450	5.444	59
1,930	3.150	3.162	17	2,165	4.325	4.323	44	2,400	5.475	....	....
1,935	3.175	....	....	2,170	4.350	....	....	2,405	5.500	....	....
1,940	3.200	3.195	18	2,175	4.375	4.383	45	2,410	5.525	....	....
1,945	3.225	3.228	19	2,180	4.400	....	....	2,415	5.550	5.553	60
1,950	3.250	....	....	2,185	4.425	....	....	2,420	5.575	....	....
1,955	3.275	3.263	20	2,190	4.450	4.448	46	2,425	5.600	....	....
1,960	3.300	3.296	21	2,195	4.475	....	....	2,430	5.625	....	....
1,965	3.325	3.329	22	2,200	4.500	....	....	2,435	5.650	5.639	61
1,970	3.350	3.362	23	2,205	4.525	4.513	47	2,440	5.675	....	....
1,975	3.375	....	....	2,210	4.550	....	....	2,445	5.700	....	....
1,980	3.400	3.395	24	2,215	4.575	4.578	48	2,450	5.725	....	....
1,985	3.425	3.432	25	2,220	4.600	....	....	2,455	5.750	5.745	62
1,990	3.450	....	....	2,225	4.625	....	....	2,460	5.775	....	....
2,000	3.475	3.471	26	2,230	4.650	4.643	49	2,465	5.800	....	....
2,005	3.500	3.510	27	2,235	4.675	....	....	2,470	5.825	....	....
2,010	3.525	....	....	2,240	4.700	4.709	50	2,475	5.850	5.851	63
2,015	3.550	3.549	28	2,245	4.725	....	....	2,480	5.875	....	....
2,020	3.575	....	....	2,250	4.750	....	....	2,485	5.900	....	....
2,025	3.600	3.588	29	2,255	4.775	4.783	51	2,490	5.925	....	....
2,030	3.625	3.629	30	2,260	4.800	....	....	2,495	5.950	5.957	64
2,035	3.650	....	....	2,265	4.825	....	....	2,500	6.000	....	....
2,040	3.675	3.673	31	2,270	4.850	4.858	52	2,505	6.025	....	....
2,045	3.700	....	....	2,275	4.875	....	....	2,510	6.050	6.060	65

CLASSIFYING AND DRYING THE CONCENTRATES.

The concentrates from different concentrating units are kept as separate lots, such as hand-sorted ore, coarse-jig concentrates, fine-jig concentrates, table concentrates of different quality, and slime concentrates.

For drying the concentrates most of the mills use an ordinary heating stove or salamander made of sheet iron with a grate for fuel and a large stack (fig. 2). The stove is surrounded by a jacket perforated at the bottom. Wet concentrates charged into the jacket come in contact with the hot stove. As they are dried they drop through the perforations onto the floor, where they are allowed to cool, and are then sacked. The ashes from these stoves are held and treated, which results in a recovery of concentrate amounting to as much as 6 pounds a day.

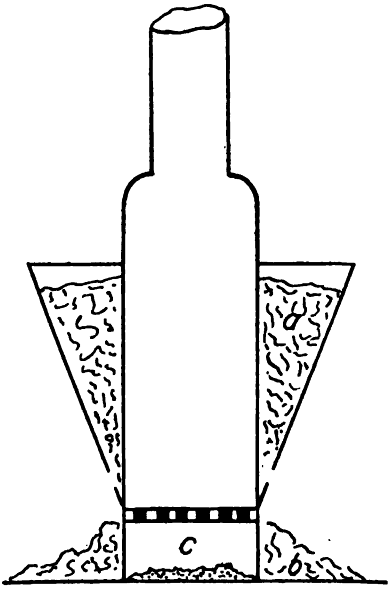


FIGURE 2.—Concentrate drier: *a*, Moist concentrate; *b*, dry concentrate; *c*, ash pit.

The mill of the Tungsten Products Co. uses steam tables for drying.

METHODS OF CONCENTRATION.

CONCENTRATION OF COARSE MATERIAL.

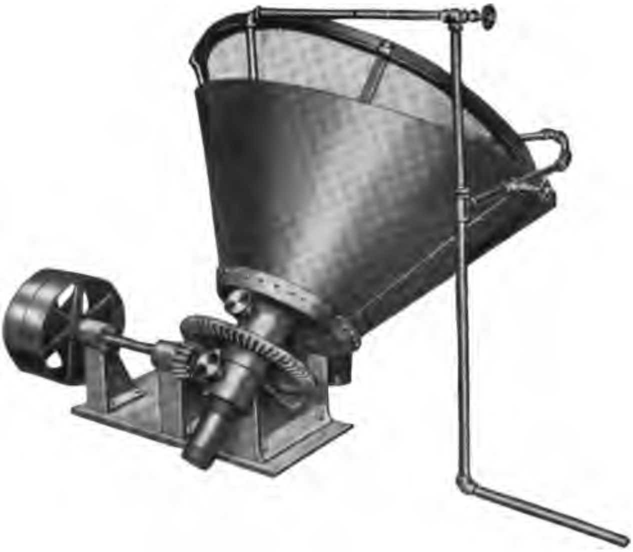
SORTING.

Hand sorting of the high-grade pieces, when practiced, is usually done in the stopes at the mine, although a small amount may be done at the mill. None of the mills, however, have any elaborate equipment for this purpose. Careful mining of the high-grade streaks, when encountered, obviates much sorting.

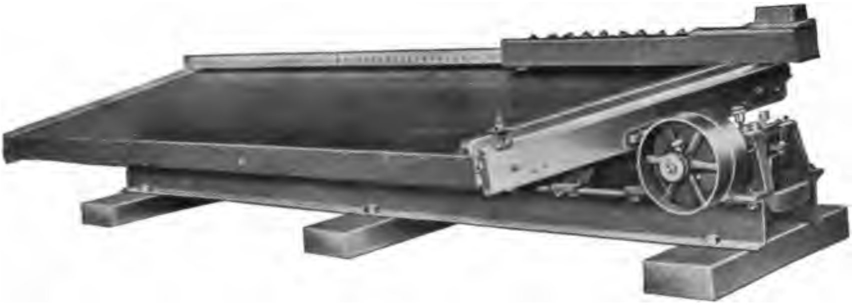
Sorting of the waste is not practicable, as exact distinction between the waste and a low-grade ore worthy of treatment is difficult. For instance, material carrying 1 per cent WO_3 may contain a large number of pieces of ore that surficially appear absolutely barren.

CRUSHING AND SCREENING.

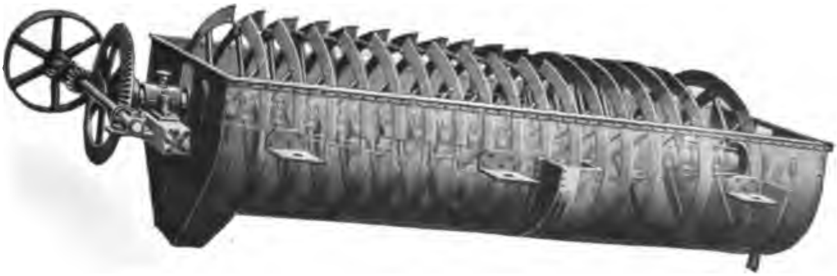
Screening begins with grizzlies having ½ to 1 inch openings. The grizzlies are set ahead of the primary crushers, which are all of the Blake type and range in size from 7 by 10 to 10 by 16 inches, the smaller size predominating. The size of the material delivered from



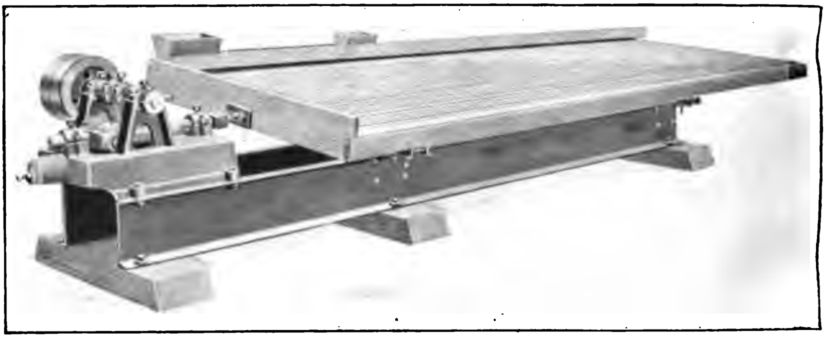
A. BUNKER HILL SCREEN.



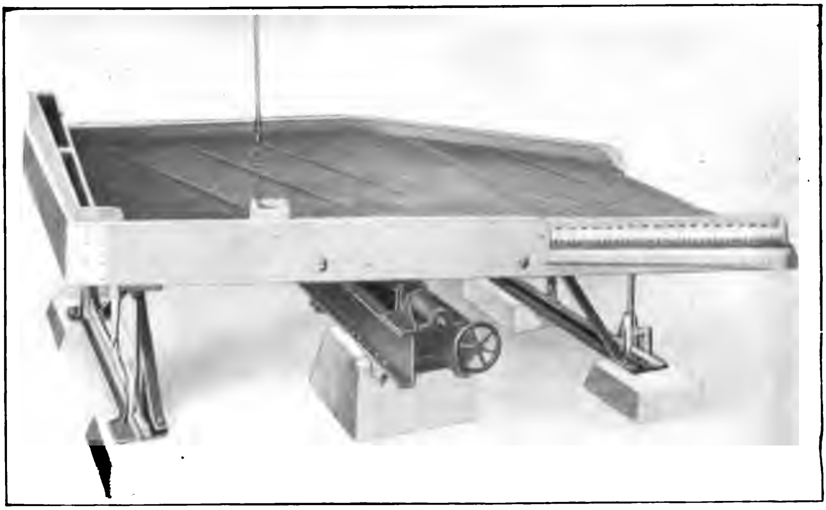
B. CARD CONCENTRATOR.



C. AKINS CLASSIFIER.



A. WILFLEY CONCENTRATOR.



B. DEISTER NO. 3 SLIMER.

these crushers ranges from 0.75 inch to 1.5 inches, but is generally 1 inch. The crusher product, which may or may not be screened, goes to rolls ranging in size from 12 by 20 to 16 by 36 inches, the usual size being 14 by 27 inches. Practice is about evenly divided as regards screening the crusher product, but where it is done trommels having from 4 to 8 mesh screens are used. In the Vasco mill the product from the primary crusher is sent to a secondary crusher with a stationary screen intervening, and thence direct to the primary rolls. The final roll product is generally about 4 mesh. The feed is then ready for sampling and storing in the settlement bins. All crushing and screening are done dry up this stage.

The next step is to screen by wet trommels having screens of 4 to 8 mesh, the oversize going to secondary rolls and the undersize, with or without further sizing by trommels, to coarse jigs. Jig tailings are then reground by rolls, except at the Wolf Tongue mill, which uses stamps.

Where the material is screened before sampling, trommels are run without water. For sizing the jig feed the trommels and impact screens are operated wet. Bunker Hill screens, ranging in size from 30 to 120 mesh, are employed extensively for preparing table feed. A view of a Bunker Hill screen is shown in Plate V, *A*.

JIGGING.

Practice in coarse jigging is about evenly divided between Harz jigs and Richards pulsating jigs. In size, the feed to the coarse jigs is about that passing a 4-mesh screen and retained by a 12-mesh screen. Screens are trommels, or of the impact type. At the mill of the Tungsten Products Co. the jig feed has been ground to pass an 8-mesh screen and the entire product is sent to the jigs without removal of the fine material.

Harz jigs and Richards launder-type pulsating jigs are used for jigging material about 20-mesh in size.

The coarse jigs make three products—concentrate, hutch, and tailing; and the fine jigs, two—concentrate and tailing. The hutch product may be retreated on jigs and tables, or it may be a finished product. Tailings are reground for sand-table feed.

Plate IV (p. 19) shows a two-compartment Richards pulsating jig.

TABLES.

For treating sands Wilfley and Card tables are used. A Card table is shown in Plate V, *B*, and a Wilfley in Plate VI, *A*.

Sand tables make three products—concentrate, middling, and tailing. The middling is held for separate treatment or reground. The tailing is classified in an Akins classifier, and the coarse is reground in ball mills.

CONCENTRATION OF THE FINE MATERIAL.

Concentration of the fine ferberite is accomplished by slime tables and stationary canvas tables, or, as they are locally known, "rag plants." The tables used are the No. 3 Deister, Deister-Overstrom, and Wilfley. The Monell and Wilfley box slime-tables that were used extensively several years ago have no place in the modern mills.

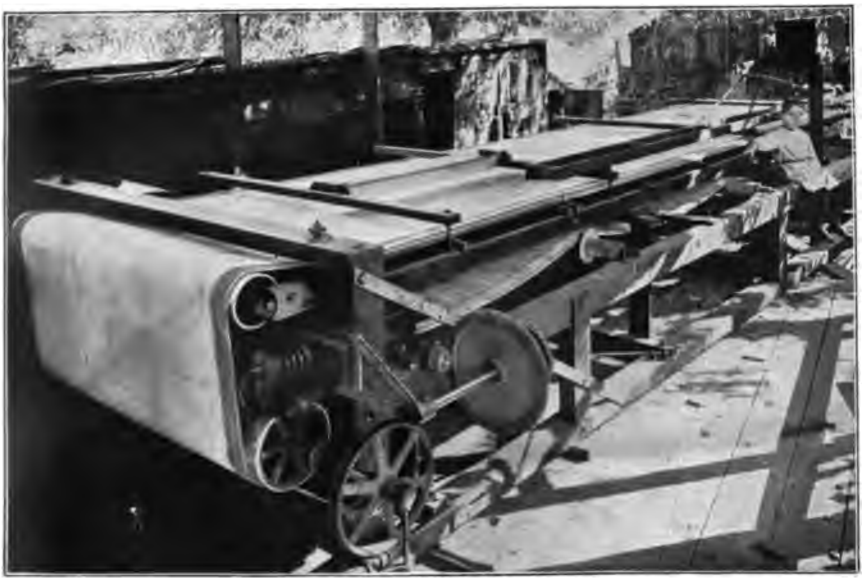
A new feature of the slime tables is a high-grade white enamel coating on the entire deck and riffle surface. Although some millmen believe this facilitates the treatment of ferberite slimes, the writers believe that it is of no value other than by affording a white background for the black mineral streak; it may help in defining the "point of cut," especially where the light is poor.

Slime tables may make two or three products—concentrate and tailing, or concentrate, middling, and tailing. The middlings may be reground and placed back in circuit, kept in a middling circuit, or held for retreatment when enough accumulates to warrant it. This, of course, is practically the same as a separate middling circuit. Tailings are generally thickened and sent to another set of slime tables, whose tailings may be further treated in this manner or sent to a "rag plant." Plate VI, *B*, shows a Deister No. 3 slimer.

The "rag plants" consist of stationary tables covered with canvas and with a slope of about 0.75 inch to the foot. They are usually arranged in parallel in a series of groups. The pulp passes over the tables with a slow, rolling movement that permits the heavy mineral to settle into the interstices of the canvas. The tables are cleaned by shutting off the feed and washing down the surface with a hose. The concentrates are collected in suitable launders placed at the ends of the tables.

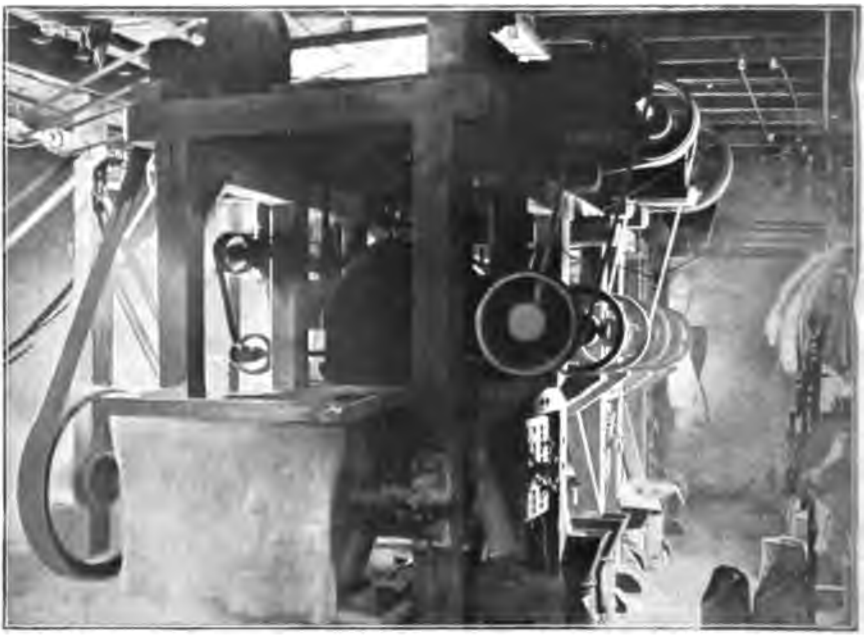
In size the tables are about 6 feet wide and 20 to 40 feet long. They require considerable mill room when treating a large feed, but are not expensive to maintain in repair when in continual operation and do not require much attention. The grade of the concentrate is low, about 20 per cent, and therefore the concentrates are retreated on slime tables.

For classifying, settling boxes making several spigot products are valuable, as they do not introduce additional water into a pulp that should be kept as thick as possible. Some upward-current classifiers are used, but the extra water necessary is carefully watched. The Akins classifier is always used for separating sand from slime for ball-mill feed. Callow cones are in favor for thickening slime for tables. Plate V, *C*, represents the type of Akins classifier seen in the district.



A. MONELL SLIMER.

B. MAGNETIC SEPARATOR ROOM OF THE TUNGSTEN PRODUCTS CO.



PART OF MAGNETIC SEPARATOR OF TUNGSTEN PRODUCTS CO.

The magnetic treatment, especially when applied to the Boulder ores, has the disadvantages of dust losses due to the necessity for dry crushing and of inability to handle economically the finer sizes, to which much of the ore of this district must be crushed in order to free the ferberite. Also, the initial cost has to be considered, although a machine of the Wetherill type is simple and well constructed and runs with little attention and at a low cost after adjustment to the requirements of the ore treated.

The only installation of magnetic separators in the district is in the mill of the Tungsten Products Co., at Boulder, where are two Wetherill type "E" No. 3 machines, which have a feed belt 18 inches wide. They have three double-pole magnets, which are wound respectively for 100,000, 60,000, and 30,000 ampere turns. Magnetite, siderite, clean ferberite, and ferberite middlings are lifted in the order named, the pyrite, phosphorus minerals, and clean gangue pass into the tailing. For concentrating fairly high-grade ores (20 to 40 per cent), in which the ferberite is liberated without excessive crushing, magnetic separators are of value with a wet concentration plant. The fine material that the magnetic separators are unable to handle may be put into the wet mill circuit. Plates VII, *A*, and VIII, *A*, give two views of the magnetic separators at the mill of the Tungesten Products Co.

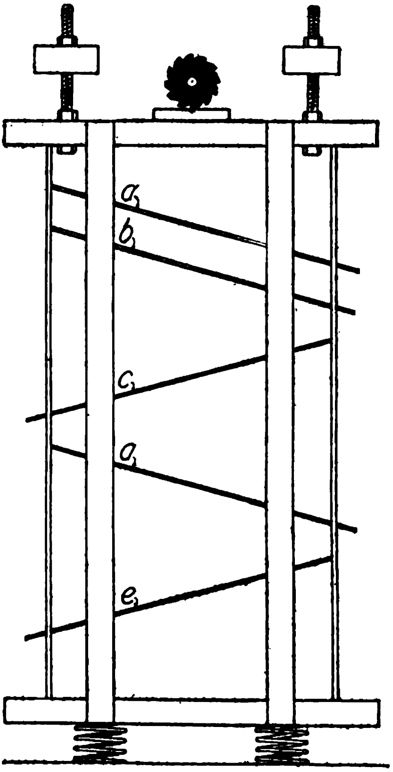


FIGURE 3.—Ellis screen: *a*, 20-mesh; *b*, 40-mesh; *c*, 60-mesh; *d*, 80-mesh; *e*, 100-mesh screen.

DRY SCREENING.

For preparing the feed to the magnets for close dry-sizing and for grading up products, screens of various types are used. At the Tungsten Products Co. mill is a screen devised by William Ellis, mill superintendent, shown in figure 3. It consists of a series of superimposed screens, 1.5 feet wide by 4 feet long, set at an angle of 14° in a suitable dust-proof housing. Vibration is imparted by a ratchet with 12 teeth, revolving 125 revolutions per minute, and imparting a total of 1,500 impacts a minute to the frame. The ratchet is keyed to a revolving shaft that carries the driving pulley. The tension upon the springs may be easily adjusted. Suitable chutes convey the screen products into separate bins. Another type of screen, called the Allen, that has been used in the district, utilizes several weighted pulleys attached to the driving shaft for imparting vibrations to the screens, which are supported in a frame by hickory legs or on springs. The Bland screen, described in another part of this bulletin under "The Grimm Concentrating Mill," is much like the Allen screen, but differs in the vibrating mechanism.

LOSSES.

The concentration loss in treating ferberite ores is due almost entirely to the difficulty of treating the slimes. The standard tables make a satisfactory high-grade product from the coarser sizes, about 20 to 80 mesh, and jigs from the larger sizes. The tailings from these devices vary greatly in value according to the nature of the ore treated. For ores in which the ferberite is coarsely crystallized, the tailings are low, as the valuable mineral is usually well liberated from the gangue; but in the "hornstone" ores the ferberite is so intimately associated with the quartz that a great amount of crushing is needed and a high tailing is inevitable. The amount of high-grade concentrates produced in treating the latter type of ore will be very low.

The ferberite is so much more brittle than the accompanying gangue that in crushing it is reduced to particles much smaller than the gangue particles, and although its specific gravity is higher, it is acted upon by the same forces and in the same way as is the gangue. The presence of the smaller size of the ferberite particles was demonstrated to the writers at the mill of the Rare Metals Ore Co., at Rollinsville, in observing the work of the Bland screen. (See p. 37.) The feed to the machine was a rag plant concentrate, carrying about 20 per cent WO_3. Three products were made on calibrated China silk fabric, as follows: Material retained on cloth with 0.004-inch openings (corresponding to Tyler screen 150 mesh); minus 0.004-inch plus 0.002-inch material, and minus 0.002-inch material (corresponding to Tyler screen 280 mesh). The latter product was very different in appearance from the feed, being black and resembled high-grade slime concentrate. According to S. B. Tyler, superintendent of the mill, this product usually assays between 40 and 50 per cent WO_3. Deposition of these products is described on page 42.

The slime losses may easily amount to 20 to 30 per cent of the valuable mineral content of the original ores, and where much "hornstone" is present, the losses may be larger. Unless slime tables or a "rag plant" is used the total recovery will be lowered. A "rag plant" or a set of slimers will save between 10 and 20 per cent of the tungsten in the original ore.

The results of the following screen analysis of a typical sample of tungsten-mill tailing shows how sliming effects losses:

TABLE 5.—*Distribution of tungsten losses in screen products from a typical sample of tungsten-mill tailing.*

Mesh.	WO_3 content.	Loss.
	Per cent.	*Per cent.*
20–40	0.63	2.0
40–60	.57	6.9
60–80	.375	7.0
80–100	.365	2.0
100–200	.275	10.1
Minus 200	1.41	66.0
	....	100.0

DESCRIPTIONS OF INDIVIDUAL PLANTS.

THE TUNGSTEN PRODUCTS CO.

The Tungston Products Co. owns and works the Luckie 2 mine in Boulder Canyon (Middle Boulder Creek), about 9 miles above the town of Boulder. In Boulder the company has a concentrator, a plant for making tungstic oxide, and a smelter for producing ferro-alloys, the company handling custom ore as well as ore from its own properties. This is the most complete plant for mining and treating ferberite ores in the district, as it embraces all phases of tungsten production. The company has an excellent laboratory with facilities for research as well as for routine analytical work. The mill circuit is shown in figure 4.

This diagram and the others in this bulletin were made by the writers during their visit to the district in January, 1919. None of the mills visited had the graphic flow sheets brought up to date, so it was necessary to study out the circuit for each mill.

A striking feature of this concentrator is the excellent work done by the Harz jigs. These jigs (fig. 5) have been modified by the mill superintendent, William Ellis, according to the following ideas:

The partition between the plunger compartment and the screen compartment is arranged so that both compartments have the same width. This may be done either by moving the partition or by building up the sides. In the screen compartment the sides below the screen supports are filled in so that they are flush with the supports. In this way a more even pressure is exerted on the bed of the jig. The screen is of the same mesh as the trommel, whose undersize is the feed. For low-grade ores it may be necessary to use a screen with slightly smaller openings than the trommel in order to

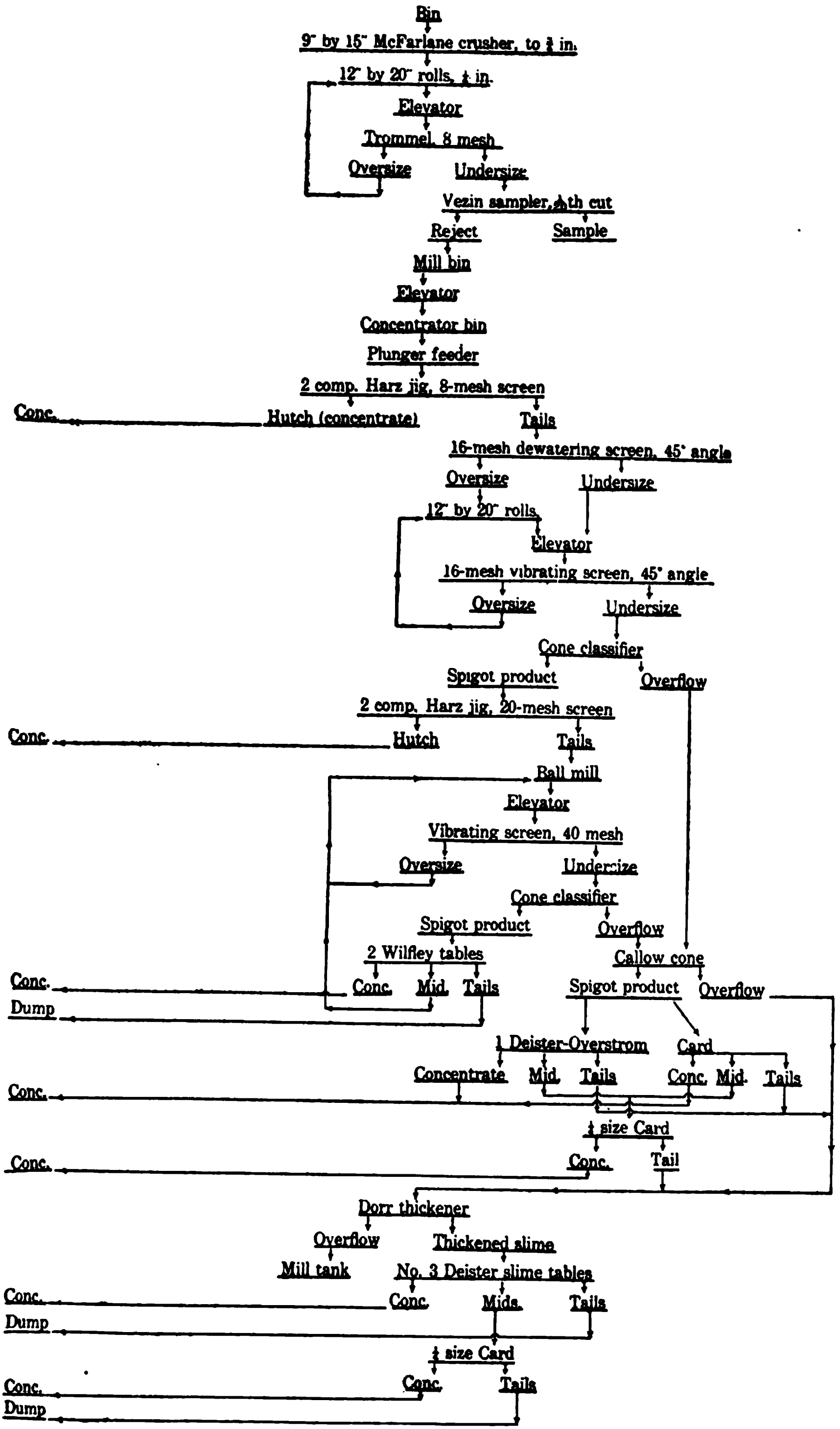


FIGURE 4.—Mill circuit of Tungsten Products Co. mill, Boulder, Colo.

keep the bed constant. On 8-mesh material the jig is run with a throw of ¼ to ⅝ inch at a speed of 250 strokes a minute.

The jigs make 60 to 70 per cent of the total concentrate produced from the average ore, the product assaying 60 per cent or more tungstic oxide.

As the diagram shows, the undersize from the first 16-mesh dewatering screen passes to an elevator, joins the oversize that has been crushed, and goes with it to another 16-mesh screen. This peculiar flow is necessary on account of the mill arrangement. The screens are a modified type of the Colorado impact screen and are made in local shops.

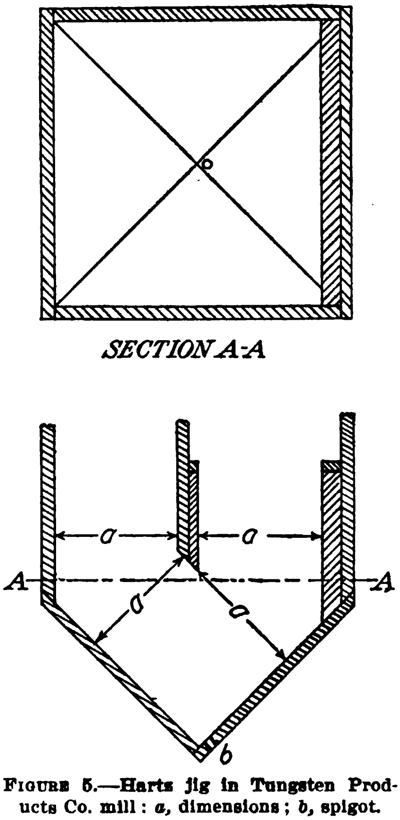


FIGURE 5.—Harts jig in Tungsten Products Co. mill: *a*, dimensions; *b*, spigot.

The middling product from the Deister-Overstrom and Card tables is re-treated on a quarter-size Card table without crushing. This procedure makes a rather low-grade concentrate but avoids the losses incident to further crushing, as does the treatment of the middling from the Deister No. 3 slime table.

A dust collector is connected to the dry-crushing equipment.

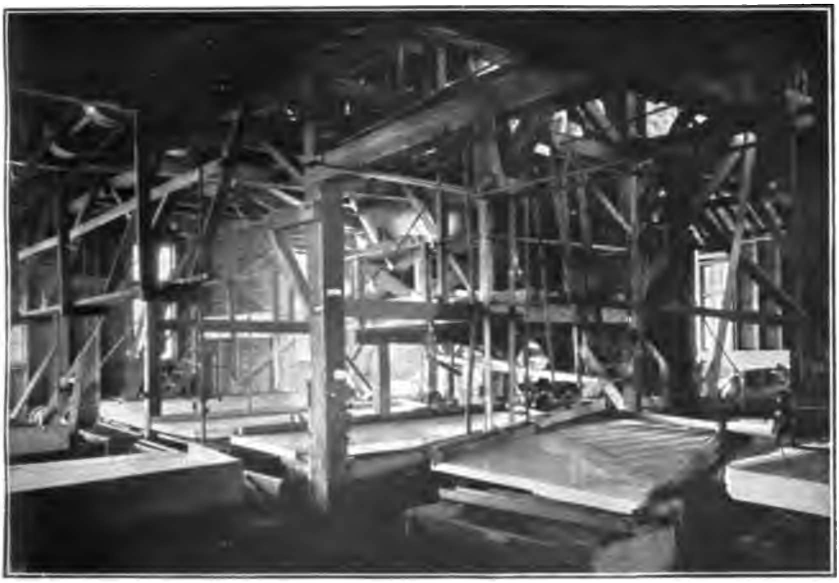
Different lots of crude ore may be retained in the mill bins until settlement for them is made, when they are transferred to the concentrator bin for treatment.

The high-grade concentrates from jigs, the tables, and the "slimers" are sent direct to the ferrotungsten plant. The second-grade concentrates are treated by Wetherill magnetic machines to remove impurities and to raise the grade. Plate IX, *A*, shows the exterior of the concentration mill of the Tungsten Products Co., and Plate IX, *B* (interior view), shows the concentrating tables.

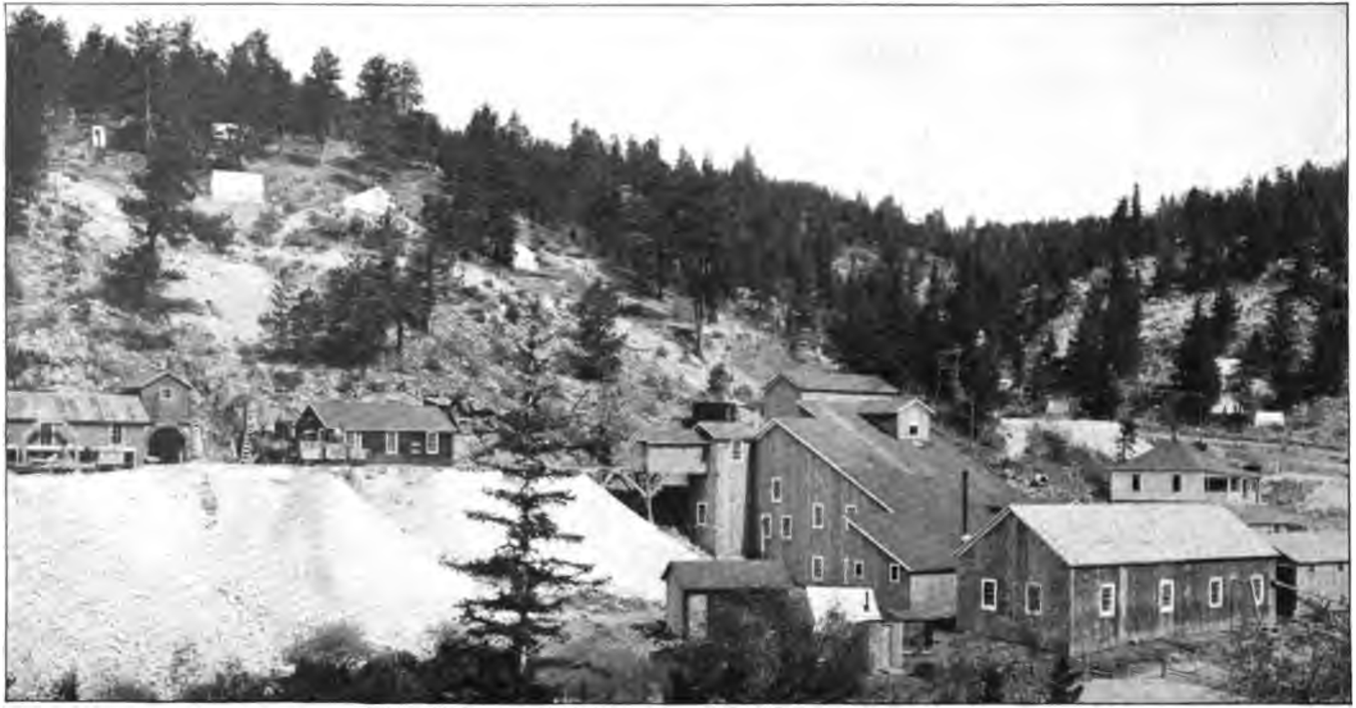
THE GRIMM CONCENTRATING MILL.

The Grimm concentrating mill is about 7 miles up Boulder Canyon (Middle Boulder Creek), near Ferberite, Colo., and is experimental in size, as only 1,000 pounds of material can be treated in one shift. The mill is operated by George Bland to demonstrate the "Bland

A. TUNGSTEN PRODUCTS MILL, TUNGSTEN, COLO.



B. TABLES IN MILL OF TUNGSTEN PRODUCTS CO.



MILL OF THE BOULDER TUNGSTEN PRODUCTION CO., TUNGSTEN, COLO.

screen," which was invented by his brother, John Bland. Claims made for this screen are that it will treat dry ball-milled tungsten ores and with some ores produces a direct concentrate, whereas with others the screened sized products will give better recoveries and cleaner concentrates when submitted to the tables than when classified by other means. The method of treatment practiced in 1919 was as follows:

Dry crushing in a ball mill, dry screening on the Bland screens of 40 and 60 meshes; the +40 material going to one Wilfley table, the —40 to +60 to another Wilfley table, and the —60 to Deister tables. In 1919 nothing was being done with the middling, tail, or slime. The middling and tail collected presumably will be dried and reground; the slime goes to waste.

The plant did not seem to be complete, at the time it was visited (January, 1919), as no jigs nor "rag plant" were installed.

The Bland screen has four superimposed screens, about 2 by 5 feet each, attached to a frame having four especially constructed legs of hickory wood. The whole frame is vibrated by a steel-tooth ratchet with 12 teeth that revolves about 300 times a minute, giving 3,600 impacts a minute to the frame and screens. Various sized China silk screens can be employed, with openings up to 0.0013 inch in diameter (corresponding to a hypothetical Tyler screen of 350 mesh), the object being to dry-size the material closely.[18]

In many of the concentrating plants in the district this screen had been installed, but later, on account of its small daily capacity, had been discarded, except in the Rare Metals Ore Co. mill at Rollinsville, Colo., which is described in connection with the treatment of the "rag plant" concentrates.

THE RED SIGN MILL.

The Red Sign mill is at Ferberite, about 7 miles up Boulder Canyon (Middle Boulder Creek) from Boulder, and is compact and well designed. Careful stage concentration is a feature, beginning with Richards pulsating jigs for material between 5 and 12 mesh in size. Bunker Hill screens of 30 and 50 mesh are used, and are said to be satisfactory. All tables make but two products—concentrates and tailings. Callow cones are used for thickening the slime table feed. The capacity of the mill is about 25 tons a day when treating 1 per cent ore. Cherty and rebellious hard silicates are treated. The flow sheet is similar in principle to those of other mills in the district but differs from that of the Tungsten Products Co. mill in that Richards jigs and Bunker Hill screens are used in preference to Harz jigs and impact screens. Treating 1 per cent ore the mill was making a saving of about 85 per cent. The concentrates are not all of 60 per cent grade, as some lower grade material can be shipped to the eastern refining plants for manufacture of tungstic acid and other products. The mill circuit is shown in figure 6.

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[18] Bland, George V., Dry sizing as a means of preparing feed for concentration: Eng. and Min. Jour., vol. 105, May 18, 1918, p. 908; Bland, John, Notes on screening: Eng. and Min. Jour., vol. 107, No. 26, June 28, 1919, p. 1112.

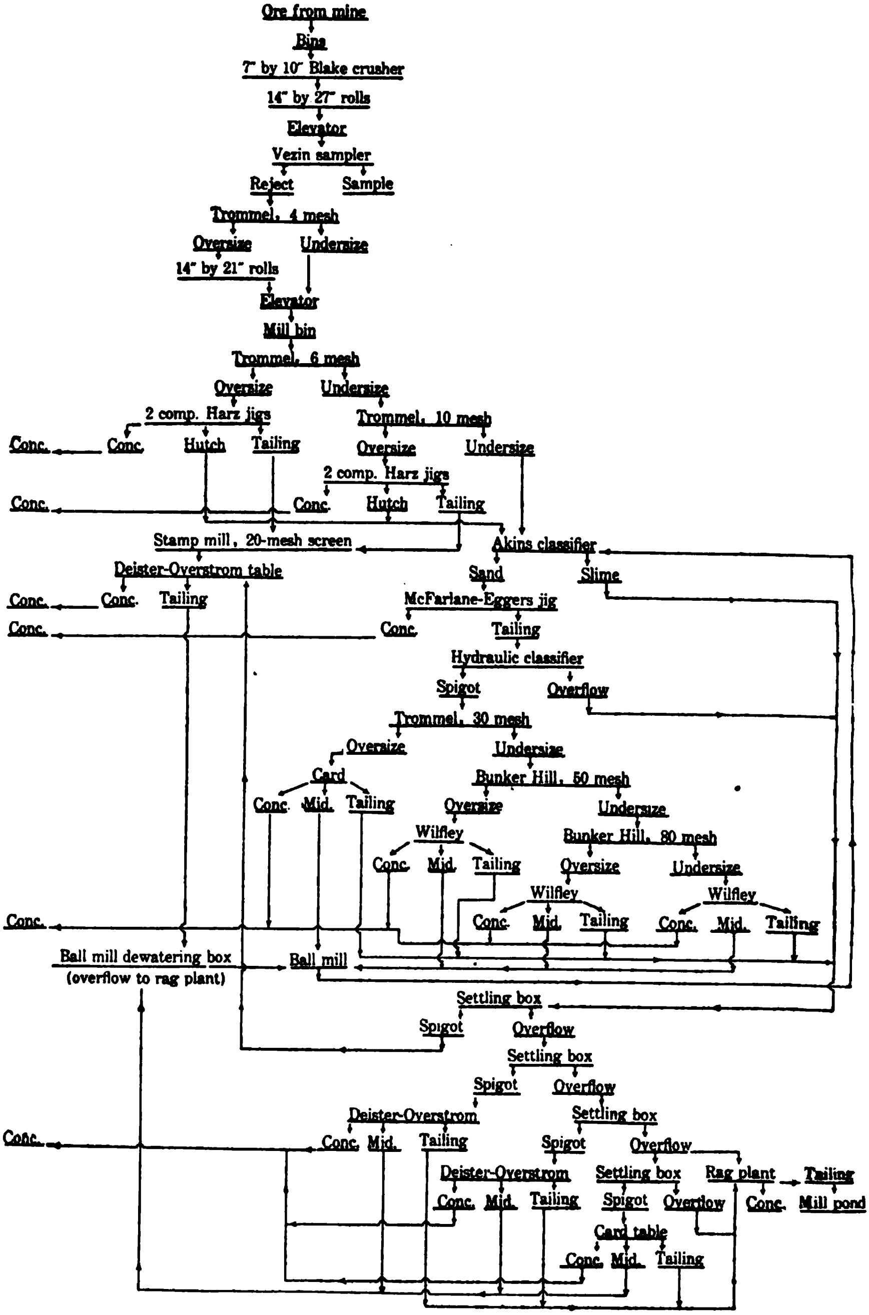
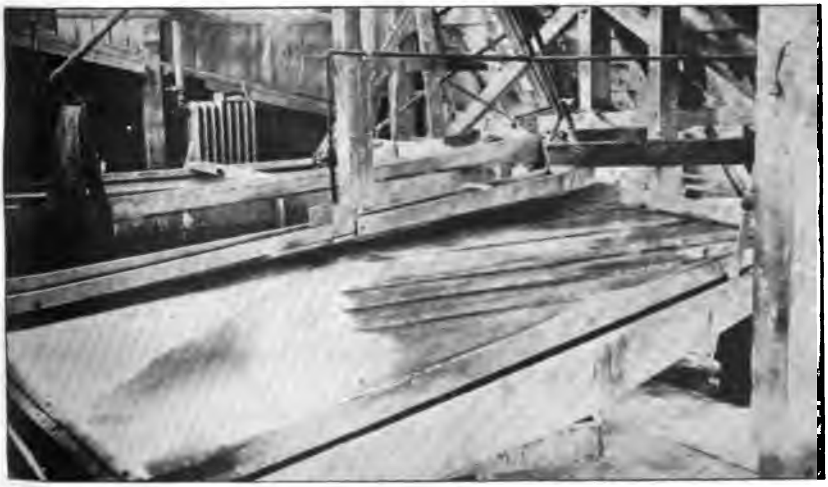


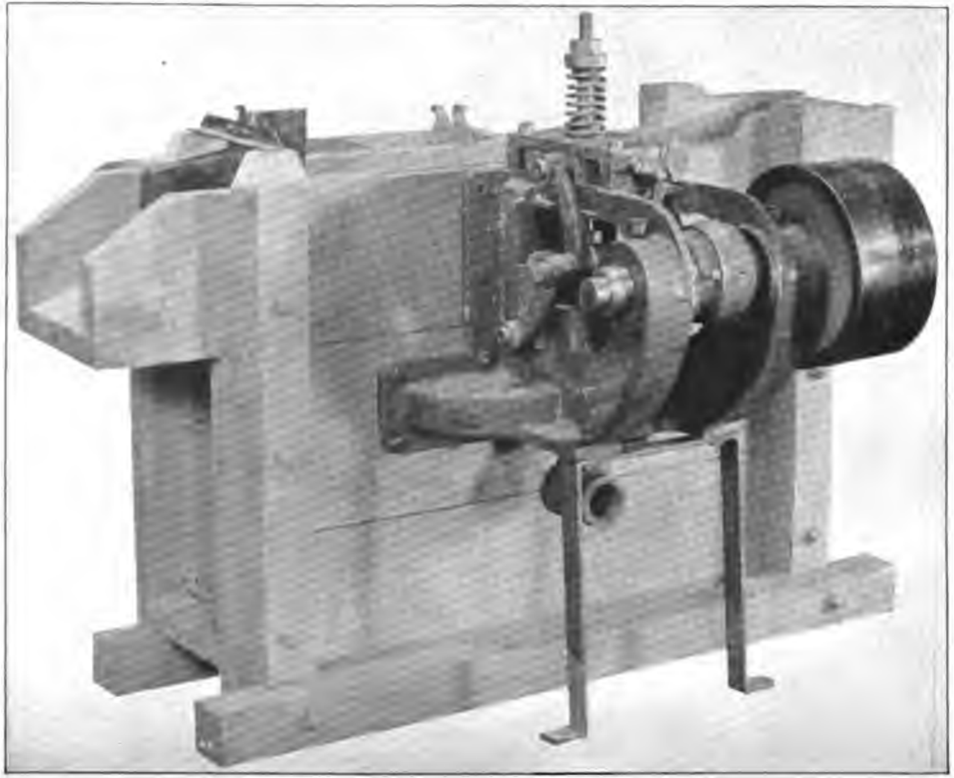
FIGURE 6.—Mill circuit of the Red Sign mill, Ferberite, Colo.

A. THE WOLF TONGUE MILL.

B. THE VASCO MILL.



C. DEISTER-OVERSTROM TABLE WITH WHITE ENAMELED DECK.



EGGERS DIFFERENTIAL PULSATING JIG IN USE AT THE WOLF TONGUE MILL.

THE VASCO MILL.

The Vasco mill, in Boulder Canyon (Middle Boulder Creek, about 14 miles from Boulder), at Tungsten, and owned by the Vanadium Alloy Steel Co., is one of the new mills of the district and is efficiently equipped. Recoveries, according to company officials, often exceed 90 per cent and occasionally 93 per cent, based on all grades of concentrates made. The mill was built to handle about 100 tons a day of 2 per cent ore, but the general average of ore in 1919 was between 2 and 10 per cent. A large part of the ore handled was coming from the company's mines, which are numbered from 1 to 13. One of the outstanding features of this mill is the absence of a "rag plant." All slimes are treated efficiently by employing many sets of slime tables. As in the other mills, attempts were made to produce as much 60 per cent concentrate as possible, but as the company has its own refineries, lower-grade concentrates can be utilized, down to 40 per cent. Thus the total recovery is increased, as the losses incident to raising the grade of concentrates are avoided. The mill circuit is shown in Plate XIII. Plate XI, *B*, shows an exterior view of the mill, and Plate XI, *C*, shows a view of a Deister-Overstrom table with a white enameled deck.

THE BOULDER TUNGSTEN PRODUCTION CO.

At Tungsten (Stevens Camp), Colo., also is the concentrating mill of the Boulder Tungsten Production Co. As the mill was closed at the time of the author's visit, it was not studied in detail. A mill circuit was, however, made and is shown in figure 7. An exterior view of the mill is shown by Plate X.

THE WOLF TONGUE MILL.

The Wolf Tongue mill at Nederland, on Middle Boulder Creek, at the head of Boulder Reservoir, 17 miles from Boulder, Colo., is one of the old stamp mills erected to treat gold and silver ores from the Caribou district and was later remodeled to handle tungsten ores. In 1919 it was the only tungsten mill in the district that used stamps; these had been discarded as primary crushers and were used for regrinding jig tailing. The capacity of the mill was 500 to 2,000 pounds of concentrate a day, depending upon the character of the ore. About 90 per cent of the ore handled was from the company's own mines, and of this amount about one-half was mined on company account and one-half by lessees. The mill circuit is shown in figure 8 and a view of the mill is given in Plate XI, *A*.

This mill has a new type of jig, the Eggers differential-stroke pulsating jig, which has efficiently treated tungsten ores. This jig is shown in Plate XII.

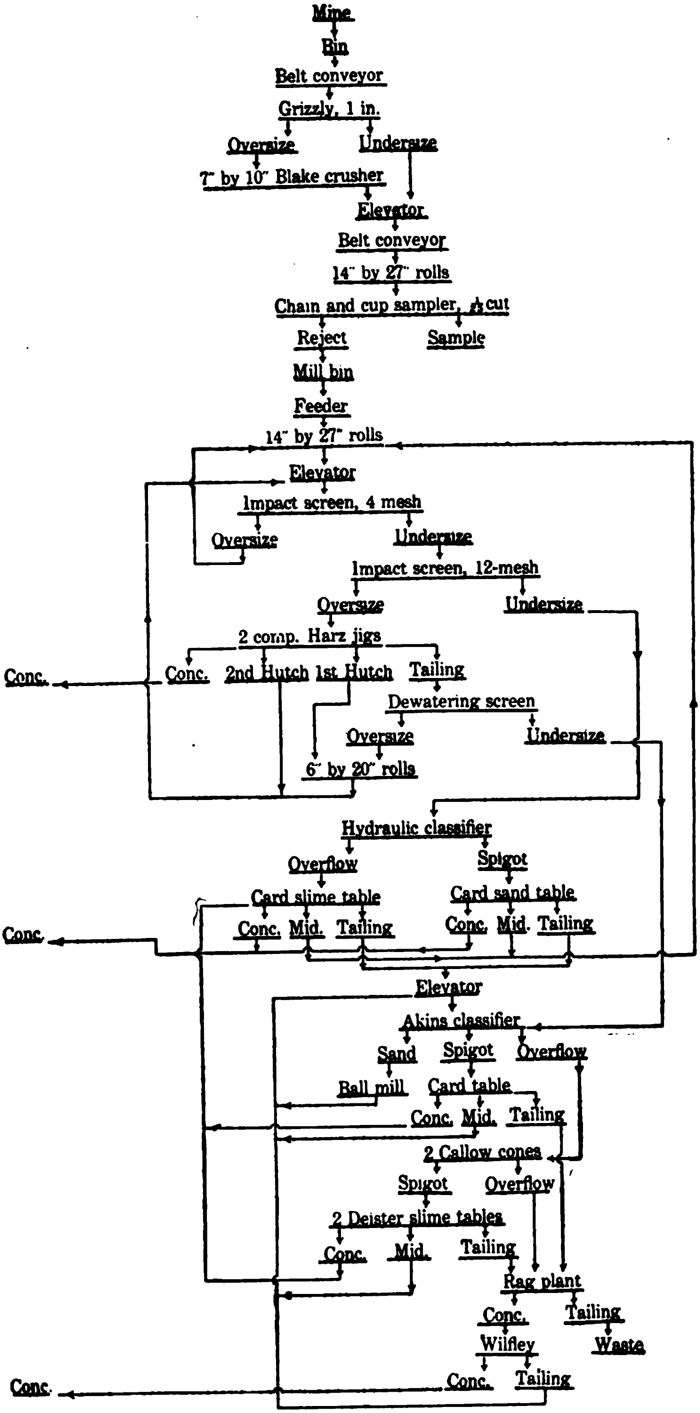
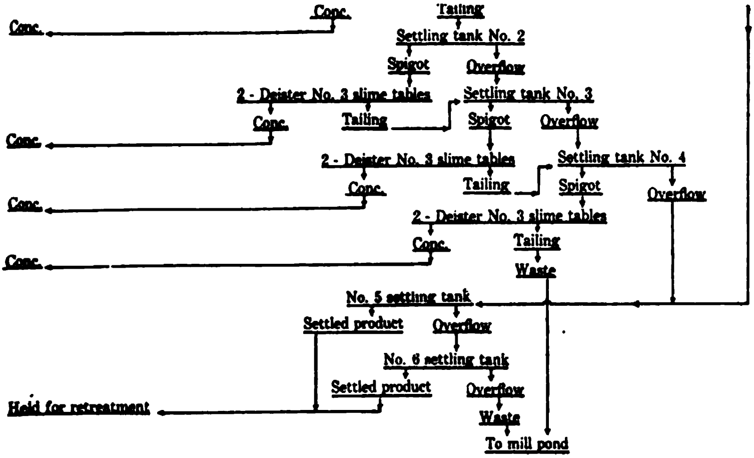


FIGURE 7.—Mill circuit of the Boulder Tungsten Production Co.'s mill, Tungsten, Colo.



THE VASCO MILL CIRCUIT.

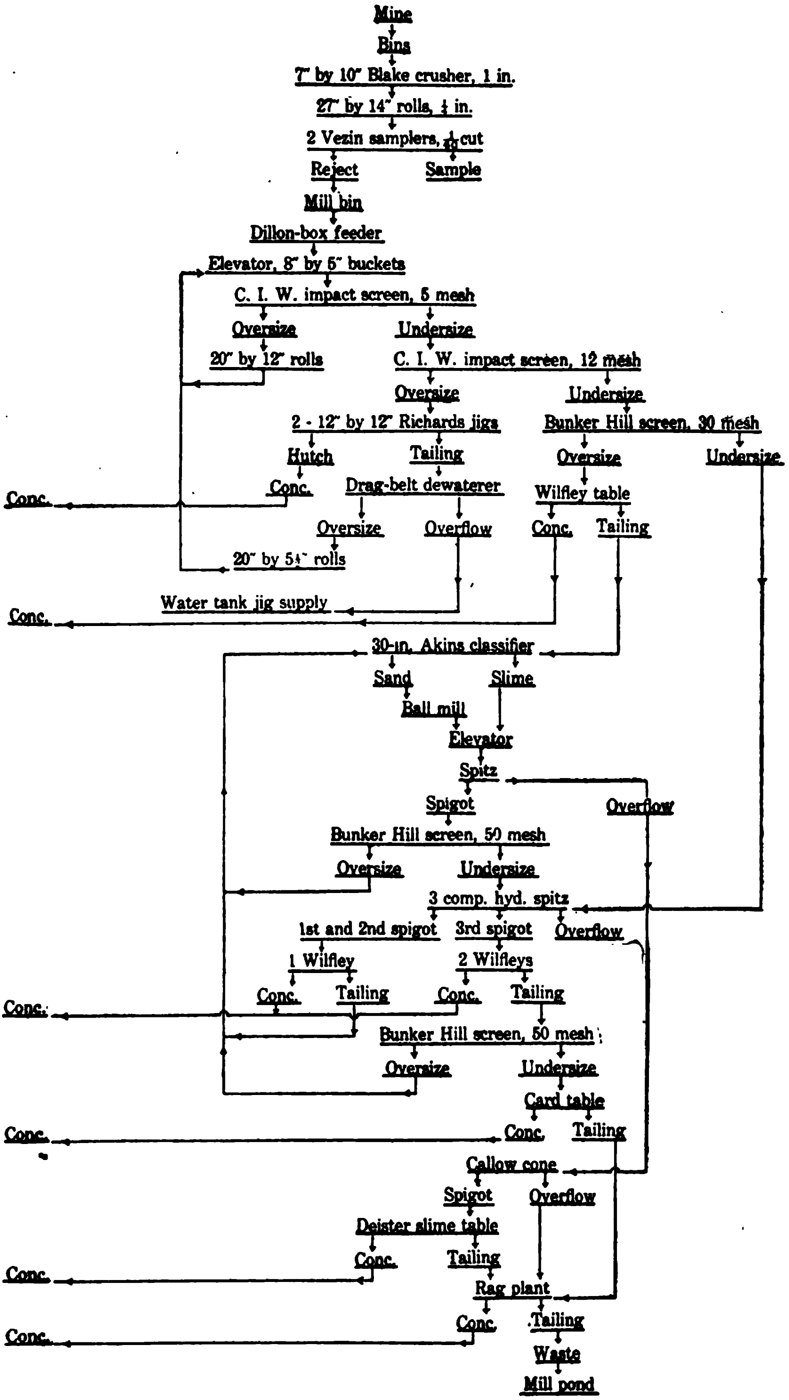


FIGURE 8.—Mill circuit of the Wolf Tongue Mining Co. mill, Nederland, Colo.

THE RARE METALS ORE CO. MILL.

The mill of the Rare Metals Ore Co., at Rollinsville, about 4 miles south of Nederland, on the Denver & Salt Lake Railroad (Moffat Road), was designed to treat ores from the company's mines in the northern end of Gilpin County, but it also handles custom ores. It is one of the new mills of the district and represents the latest in tungsten mill design. Excellent recoveries are made, even when treating the " horn-stone " ores. It has a capacity of about 50 tons of 1.5 per cent ore, or 25 tons of 4 or 5 per cent ore, or, in terms of units, from 75 to 125 units in 24 hours. Ores as low as 1 per cent are treated. Recoveries range from 80 to 90 per cent. All concentrates marketed contain 60 per cent or more WO_3.

The jig concentrate is obtained by skimming the bed at suitable intervals. The jig screens are 20 to 25 mesh.

The treatment of the "rag plant" concentrates is of interest. These concentrates, which contain about 20 per cent WO_3, are dried and go to a Bland screen, which makes three products, as follows: Material retained on 0.004-inch opening, China silk cloth, which is put in the Wilfley table feed; material passing the 0.004-inch openings and retained on 0.002-inch openings, which is fed to a separate Deister table; and material passing through the 0.002-inch openings which is high enough (40 or 50 per cent) in WO_3 to be mixed with concentrates carrying more than 60 per cent WO_3. This mill has concentrated scheelite ores, rebellious ore containing barite which was sent from the Orient for concentration, and also rejects from some of the other concentrating mills. The mill circuit is shown in figure 9, and a view of the mill in Plate XIV.

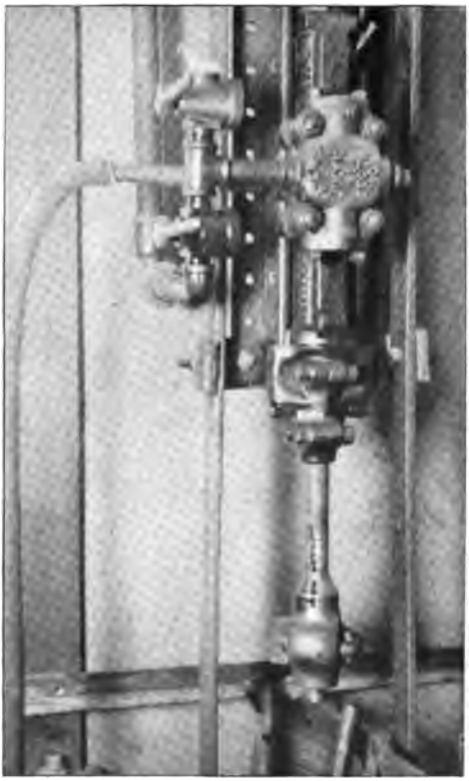
THE PRIMOS CHEMICAL CO.

The Lakewood mill of the Primos Chemical Co., at Lakewood. Colo., 4 miles north of Nederland, formerly crushed all the ore by stamps and contained no jigs, tables and the " rag plant " being used entirely. In 1919, however, practice had been changed to conform to the better methods of the other mills. The mill handles custom ore and that from the company's own properties. An exterior view of the mill and the surrounding tract, showing the Conger and Beddig mines, appears in Plate III, *C* and *D* (p. 18).

MILL REPORT SHEETS AND CALCULATIONS.

Recovery is best calculated by comparing the number of pounds of WO_3 in the heads with the total number of pounds in the different concentrate products. All figures must be based on dry weight and accurate samples. Tailing samples, judged from the samples from

PLANT OF THE RARE METALS ORE CO., ROLLINSVILLE, COLO.



A. PISTON DRILL USED AS AN AIR HAMMER FOR PULVERIZING FERROTUNGSTEN IN A STEEL MORTAR.

B. CHEMICAL PLANT OF THE BLACK METAL REDUCTION CO.

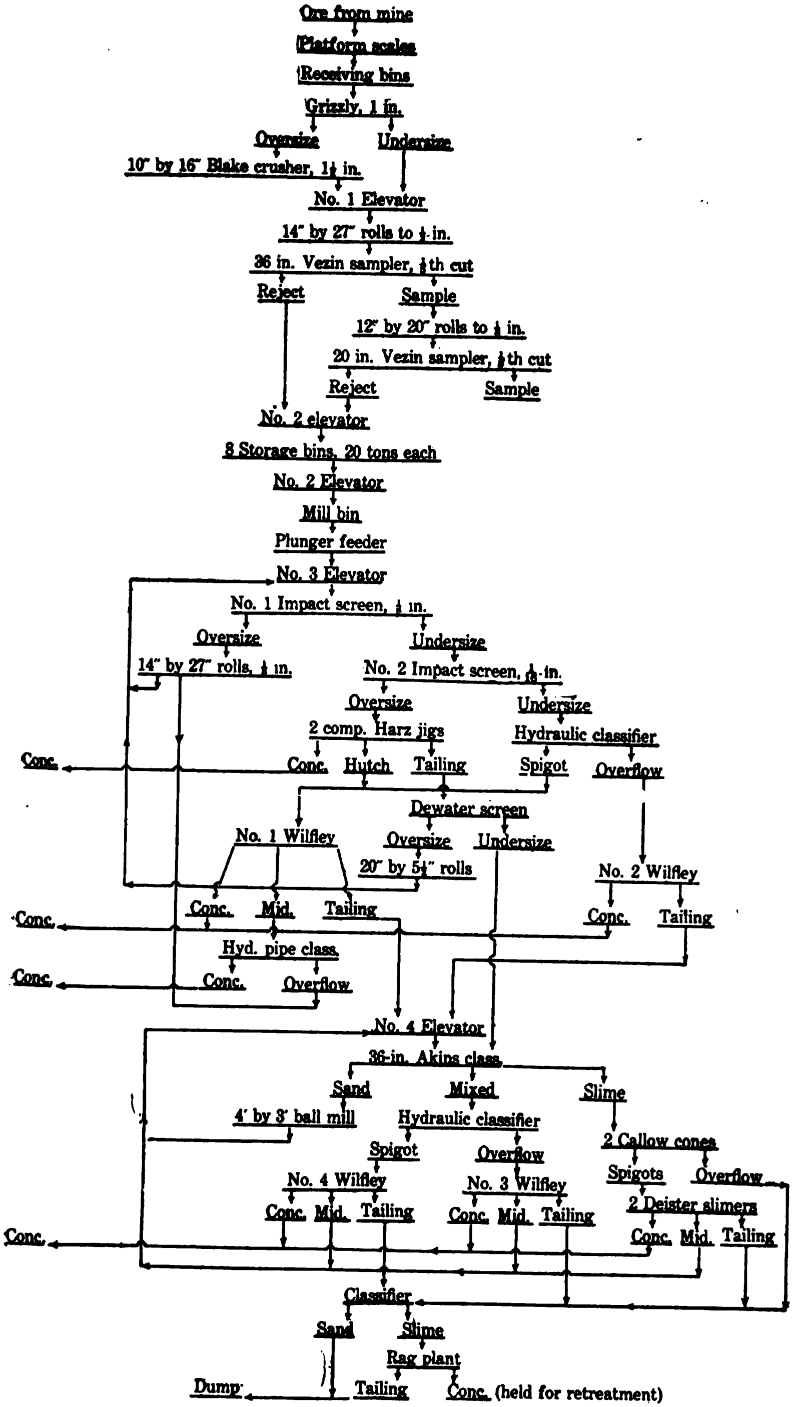


FIGURE 9.—Mill circuit of the Rare Metals Ore Co. mill, Rollinsville, Colo.

most of the mills in this district, give inaccurate results, as before the pulp passes over the rag plants, its last treatment, it is classified, the coarse going to waste and the slimes going to the canvas, thence to waste; and the two waste streams can not be combined so as to give accurate samples. In some mills where a "rag plant" is not used, settling boxes are installed for trap material which is held for retreatment later, this material may consist of thickened pulp from one or more lots, as it is not economically possible to keep each lot of such pulp separate.

At one mill—that of the Tungsten Products Co.—the mill flow is such as to give a very good tail sample, which is useful as a check. An automatic tailing sampler, as shown in figure 10, is used at this mill.

The typical mill sheet, shown in Table 6, shows how recoveries are calculated.

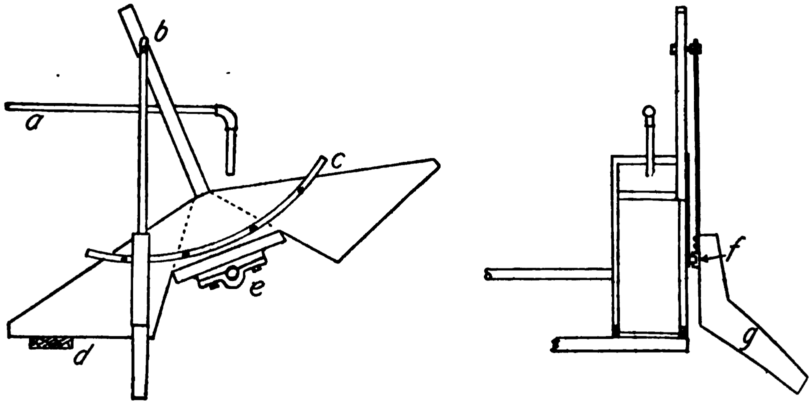


FIGURE 10.—Automatic tailing sampler: *a*, water; *b*, pivot; *c*, strap iron; *d*, stop; *e*, bearing; *f*, roller; *g*, cutter.

TABLE 6.—*A typical mill sheet.*

[Mill report run No. 56—Started Nov. 16; closed Dec. 29, 1918. Days' running time, 44, Boulder, Colo.]

Bill No.	Net weight.	Assay.	WO₃.	Gross value.	Remarks.
	Pounds.	*Percent.*	*Pounds.*		
1851	13, 323	9. 30	1, 239. 0	$712. 42	$18+15 per cent.
1852	7, 970	9. 70	773. 1	444. 53	Do.
1854	30, 144	12. 20	3, 677. 6	2, 114. 62	Do.
1855	2, 392	31. 50	753. 8	476. 77	Do.
1856	2, 688	21. 90	588. 7	338. 50	Do.
1857	1, 227	22. 10	271. 2	155. 94	Do.
1858	842	6. 20	52. 2	23. 28	Do.
1859	108	44. 40	48. 0	33. 12	Do.
1861	20, 247	11. 70	(?)2,368.9	1, 362. 11	Do.
1864	(?)277	19. 10	443. 4	25. 05	Do.
1865	40, 158	9. 65	3, 875. 2	2, 228. 24	Do.
1866	9, 638	3. 15	303. 6	87. 28	Do.
1867	824	7. 20	59. 3	34. 09	Do.
1868	10, 956	2. 55	279. 4	80. 32	Do.
1869	2, 351	6. 80	159. 9	86. 79	Do.
1873	13, 134	10. 00	1, 313. 4	755. 20	Do.
1874	921	11. 00	101. 3	58. 24	Do.
1876	6, 350	15. 00	952. 5	428. 62	$9 unit.[1]
1875	4, 156	. 85	35. 3	....	Do.
1872	5, 297	1. 05	55. 6	....	Do.
1877	4, 944	15. 90	786. 1	353. 74	Do.
1878	2, 203	33. 90	746. 8	336. 06	Do.
1879	16, 815	14. 30	2, 404. 5	1, 082. 02	Do.
1880	14, 716	15. 70	2, 310. 4	1, 039. 68	Do
	211, 632	10. 96	23, 199. 2	12, 256. 62	
Recovered	34, 592	59. 10	20, 442. 4	21, 157. 87	
Thrown away	177, 040	1. 56	2, 786. 8	....	

[1] Drop in price caused by the armistice.

CLASSIFICATION OF MILL ORES.

Under—	Net weight.	Assay.	WO₃.	Saving.	Low WO₃.	High WO₃.
	Pounds.	*Per cent.*	*Pounds.*	*Per cent.*	*Pounds.*	*Pounds.*
2 per cent	9, 453	0. 96	90. 9	40	18. 1	18. 2
2 to 5 per cent	20, 594	2. 83	583. 0	50	116. 6	174. 9
5 to 10 per cent	78, 602	9. 50	7, 472. 1	93	2, 241. 6	4, 707. 4
10 to 20 per cent	94, 364	13. 40	12, 644. 7	93	3, 793. 4	7, 966. 1
20 to 30 per cent	3, 915	21. 96	859. 9	94	171. 9	636. 3
30 to 40 per cent	4, 704	32. 92	1, 548. 6	96	309. 7	1, 176. 9
	211, 632	10. 96	23, 199. 2	91. 9	6, 651. 3	14, 679. 8

Computed from an "ultimate" table.

RECOVERED.

Bill No.	Kind.	Number of sacks.	Weight.	Assay.	WO₃.	Percentage WO₃ saved.	Value, $18 +15 per cent.
			Pounds.	*Per cent.*	*Pounds.*		
662	Jig	24	2, 280	57. 90	1, 320. 1	6. 5	$1, 366
663	do	22	2, 090	59. 60	1, 245. 6	6. 1	1, 289
664	Slimes	9	822	64. 10	526. 9	2. 5	545
665	Concentrates	9	883	59. 70	527. 2	2. 5	545
666	Jigs	109	10, 409	59. 30	6, 172. 5	30. 2	6, 388
668	Slimes	64	2, 966	60. 40	1, 791. 5	8. 8	1, 854
667	Jig	106	10, 072	57. 70	5, 811. 5	28. 4	6, 014
669	Concentrates	46	4, 529	60. 40	2, 735. 5	13. 4	2, 831
670	Slimes	6	541	57. 60	311. 6	1. 6	322
		......	34, 592	59. 10	20, 442. 4	100. 0	21, 157

294 hours on feed; 8 hours on repairs; 51 running sands and cleaning tanks.
88. 1 per cent, saving average 59.1 per cent grade.
71.2 per cent of recovery to jigs 57.8 per cent grade.
15.9 per cent of recovery to concentrates 60.3 per cent grade.
12.9 per cent of recovery to slimes, 60.7 per cent grade.

phosphorus and sulphur. At the time of the writers' visit (January, 1919) only the plant of the Black Metal Reduction Co. was running.

PLANT OF THE BLACK METAL REDUCTION CO.

The mill of the Black Metal Reduction Co. (Pl. XV, *B*) treats tungsten ores containing as low as 12 per cent WO_3 and produces a high-grade tungstic acid (WO_3), said to contain less than 0.03 per cent phosphorus, 0.3 per cent sulphur, and only a small percentage of silica. After drying, its purity ranges from 99 to 99.7 per cent WO_3. The production in 1919 was about 500 pounds of tungstic acid a day, which would correspond to treating 3,000 pounds of ore of about 20 per cent grade. The process is claimed to treat economically ore containing less than 12 per cent, but higher-grade ore is sought. The ore is bought from custom mills and from miners. A description of the process follows:

The ore is mixed with sodium carbonate and salt, and the charge is placed for about an hour in sheet-iron pans measuring 2 by 4 feet and 6 inches in depth, in a fire-brick furnace heated by a Case crude-oil burner. The charge is so proportioned that a glass slag is obtained. The pans are pulled out of the furnace onto an iron rack, are clamped and inverted, and the fused charge is dumped into an iron receptacle, where it cools. The charge is broken into pieces and ground by a jaw crusher to approximately one-half inch size. The crushed material is stored in a bin from which a weighed amount is charged into a large cement mixer and treated with hot-water wash liquor from a previous run.

After lixiviation the contents of the cement mixer are dumped into a screen-bottom trough, which retains the coarse residue. This residue is shoveled back into the cement mixer and washed with hot water, and this hot-water wash liquor is retained and used for the original wash in the next run. The concentrated solution which passes the screen trough goes into a sump, from which it is pumped into a desliming, baffle tank, and from there is run into another tank, where a further settling takes place. The clear solution from this tank is pumped to the top of the building to a so-called "special treatment tank." The treatment given in this tank, said to be the only secret connected with the process, removes the last traces of phosphorus, arsenic, silica, and other deleterious substances. The contents of this tank are put through a Sweetland filter press, from which the clear solution is run into a large wooden tank, heated by steam coils, in which calcium tungstate is precipitated by the addition of a solution of calcium chloride.

The precipitate is allowed to settle, scraped out and dumped into a wooden revolving drum, where it is treated with commercial hydrochloric acid. It is then allowed to settle and the solution is drawn off through a suction filter. The residue is treated twice in this manner with hydrochloric acid, and finally the whole is dumped onto the suction filters. The precipitate is tungstic acid. The acid solution, which contains the calcium chloride and calcium sulphate, is allowed to run into stone vessels, where the calcium sulphate finally crystallizes out. The clear acid solution is then drawn into barrels, lime is added, producing calcium chloride, which is allowed to settle. The residue obtained in these barrels usually contains 15 to 20 per cent tungstic oxide. This calcium chloride solution is used for further precipitations. The tungstic acid is placed in wooden pans, dried in a steam-piped oven and shipped in sheet-iron drums. The moisture content of this acid is 5 to 10 per cent. The process is covered by United States Patent 1255144, February 5, 1918.

MANUFACTURE OF FERROTUNGSTEN.

Three electric smelting plants have been built in Colorado to smelt ferberite concentrates, as listed in Table 7:[17]

TABLE 7.—*Colorado ferrotungsten plants.*

Company.	Location.	Capacity, kilowatts.	Product.
Tungsten Products Co.	Boulder	800	Ferrotungsten.
Boulder Tungsten Production Co.	do.	400	Do.
Ferro-Alloy Co	Utah Junction	2,700	300 kilowatts formerly used for making ferrotungsten; later all used for making ferrochrome.

TUNGSTEN PRODUCTS CO.

The plant of the Tungsten Products Co. is on the north bank of Boulder Creek, in the mouth of the canyon and on the outskirts of the city of Boulder. Power is supplied to the plant by the Colorado Power Co. at a nominal voltage of 13,200 volts. It is metered by the power company at a point outside the city limits of Boulder and is conducted to the reduction plant over the private line of the Tungsten Products Co., where the furnace transformers of the company transform it to a furnace voltage of from 55 to 90 volts. The furnaces are installed in a wooden frame building with corrugated-iron siding and roofing, but the transformers are in a fireproof substation.

Ferberite concentrate, other domestic ores, and imported scheelite and wolframite are smelted at this plant. Much of the concentrate is purchased in a low-grade form or contains impurities. Before smelting, this material is treated on a Wetherill magnetic separator to raise the grade and remove the impurities. Other raw materials are lime, silica, and coke.

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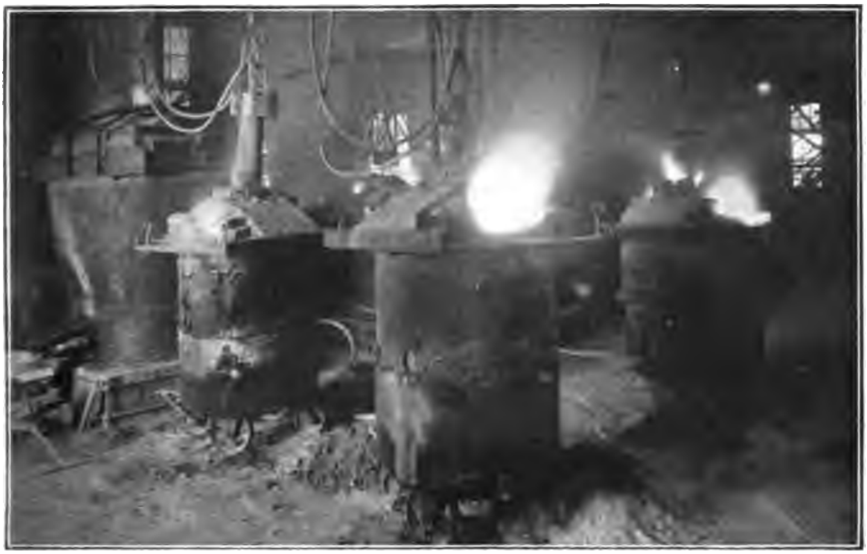
[17] Keeney, Robert M., The manufacture of ferro-alloys in the electric furnace: Bull. American Institute of Min. and Met. Eng., August, 1918, p. 1321.

The furnaces range in size from 75 to 150 kw. and have been operated at voltages ranging from 55 to 90 volts, but were finally operated at a voltage of 55 to 65 volts, as with the lower voltages metal could be produced containing less slag. The furnaces have a vertical, movable carbon or graphite electrode, a bottom contact of carbon embedded in the floor and a knockdown steel shell, lined usually with coke rammed in place with hot pitch as a binder. The so-called acid process is used, and the coke lining is used except when refining with a basic slag to remove impurities is necessary; then a magnesite lining is used. The shells of the larger furnaces are about 4 feet 6 inches in diameter by 4 feet 6 inches deep, are set on a fire-brick base, and have a turtleback roof with one end open for charging. There is a single tap hole for removal of slag as it rises in the furnace, all metal collecting in a button in the furnace bottom. The vertical electrodes are either of graphite, 4 inches in diameter, or of carbon, 6 inches in diameter, and are suspended by steel cables with automatic control of the electrodes and the load on the furnaces by Thury electrode regulators and electric winches. Plate XVI, *B*, shows the exterior of the smelter and chemical plant, and Plates XVI, *A*, and XVII show the electric furnaces in operation.

A general outline of the process follows:

The empty furnace shell, with a form in it, is set on the floor and rammed from bottom to top with coke and hot pitch. There is no bottom to the shell. When the lining has set, the shell is lifted with chain blocks and placed on the fire-brick base surrounding the bottom carbon contact. A fire-brick base is placed on the furnace, leaving one end open for charging. The electrode is lowered into place, contact through the bottom is obtained by hand control of the electrode, and charging is started by shoveling the mixture through the side roof opening at regular intervals. As soon as a bath of slag forms on the bottom the electrodes are operated automatically by the Thury regulators.

The ferrotungsten forms a pasty button about 2 feet in diameter on the lower contact and gradually builds up, raising the slag level until the latter is above the tap hole, when the slag is tapped. This continues for about 72 hours. The slag is tapped at regular intervals until the furnace contains 2,000 to 5,000 pounds of ferrotungsten, then the roof is removed, and the shell containing the metal and lining is lifted to the base. After cooling a little the lining and metal are removed from the shell. The metal button is cleaned with pneumatic cleaning tools and placed under a skull cracker of the usual steel-works type which breaks the ferrotungsten into 75 to 100



A. FERROTUNGSTEN FURNACES IN OPERATION.

B. SMELTER AND CHEMICAL PLANT OF TUNGSTEN PRODUCTS CO.

ONE OF THE LARGE ELECTRIC FURNACES OF THE TUNGSTEN PRODUCTS CO.

pound chunks. These chunks are broken to fist size under a steam hammer and crushed further in a jaw crusher and rolls to pass ¼-inch mesh.

In brief, ferrotungsten is produced at this plant in a single operation by smelting with an acid slag in an electric furnace with a carbon lining. Because of the metal being pasty as it forms in the furnace, carbon is not absorbed from the lining, and the acid slag prevents absorption from the reducing carbon in the charge. With a high phosphorus ore, however, basic refining is sometimes necessary to remove the phosphorus in the metal first reduced. As far as can be judged, the grade of product made is one of the highest produced in this country, as it averages 80 to 85 per cent tungsten, less than 0.5 per cent carbon, less than 0.03 per cent phosphorus, and less than 0.03 per cent sulphur, with about 0.5 per cent manganese and silicon. The plant has a capacity of 4,000 pounds of contained tungsten in 24 hours.

BOULDER TUNGSTEN PRODUCTION CO.

The plant of the Boulder Tungsten Production Co. is on the south bank of Boulder Creek and in the mouth of the canyon, just outside the city limits of Boulder. Power is supplied to the plant by the Colorado Power Co. at a nominal voltage of 13,200 volts and is transformed to a furnace voltage of from 50 to 100 volts by two 200 kva. furnace transformers, connected by the Scott connection. The furnaces are in a frame building.

The ore smelted was ferberite concentrate from the company's mine and mill. The plant ran only a few weeks, producing several thousand pounds of ferrotungsten of variable grade.

Four small pot furnaces of the general type described above carried a load of 75 to 100 kw. The shells were about 3 feet in diameter and 3 feet deep. The electrodes, of 4-inch graphite or 5-inch carbon, were regulated by hand-operated winches. As there was only one primary switch for the four furnaces control of a furnace was difficult. Small buttons of ferrotungsten, weighing about 500 pounds, were produced. Details of the process used were not available. This plant also includes a tungstic acid plant, which on account of the high cost of production, had been temporarily discontinued.

FERRO-ALLOY CO.

Keeney[18] has given the following description of the furnaces of the Ferro-Alloy Co.:

> The plant of the Ferro-Alloy Co., at Utah Junction, contains one 750-kw. three-phase furnace, to which power is supplied by three 250-kw. transformers,

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[18] Keeney, R. M., Work cited.

connected so as to give 129 volts on the furnace cables, the transformer ratio being 13,200 to 75 volts. The furnace operates with an actual voltage of 120 volts and produces about 3 tons of ferrochrome per 24 hours from 40 per cent Cr_2O_3 ore. The furnace consists of a steel shell of circular cross section, 8 feet (2.43 m.) in diameter by 7 feet (2.13 m.) deep, lined with magnesite, and having three vertical carbon electrodes 12 inches (30.48 cm.) in diameter.

A 450-kw. furnace, of the same size as the 750-kw. furnace, is also operated on chrome ore. This furnace is three-phase in appearance but operates electrically as a single-phase furnace. There are three vertical carbon electrodes 8 inches (20.32 cm.) in diameter with a conducting carbon bottom. Power is supplied by three 150-kw., single-phase transformers, two of which have a ratio of 13,200 to 95 volts and one a ratio of 13,200 to 100 volts. From each transformer one lead goes to one vertical electrode, and one lead to the carbon bottom, so that the transformers are electrically independent. The furnace voltage varies from 90 to 95 volts.

At intervals, 150-kw., single-phase furnaces are operated on ferrotungsten. These furnaces have one vertical graphite electrode 4 inches (10.16 cm.) in diameter, with a water-cooled steel bottom contact. The furnace shells are 4 feet (1.22 m.) in diameter and are mounted on trunnions, the slag being poured by tilting the furnace. All of the electrodes are regulated by hand. The chrome furnaces are operated without roofs and the tungsten furnaces with roofs. The whole plant has a power factor of 90 per cent.

The following description of the manufacture of ferrotungsten in Colorado has been given by Keeney:[19]

Tungsten is the leading alloy metal used in the manufacture of high-speed steel. Probably about half of the consumption is used in the form of ferrotungsten, the remainder being tungsten metal. Colorado has supplied a large part of the tungsten used in steel manufacture, but is gradually being surpassed by California, where the large scheelite deposits can produce tungsten considerably cheaper. Ferberite, found in Boulder County, Colo., is one of the most favorable forms of tungsten for production of ferrotungsten, containing both iron and tungsten in the proper proportion to yield 70 to 80 per cent ferrotungsten. At present, three electric-furnace plants in Colorado are making or have made ferrotungsten. The standard grade produced by the Ferro-Alloy Co. contains 75 per cent tungsten, 0.8 per cent carbon, 0.4 per cent silicon, 0.5 per cent manganese, 0.01 sulphur, and 0.02 per cent phosphorus. The Tungsten Products Co. makes about the same grade of product, except that the carbon is kept below 0.5 per cent and the tungsten runs as high as 85 per cent. The product of this company is probably the highest grade and the purest ferrotungsten made in this country. Ferrotungsten has a high density, fine gray fracture, and is not crystalline—the higher the carbon the coarser the fracture. The carbon in ferrotungsten occurs as the double carbide, $Fe_3C.W_2C$.

One method commonly employed for production of ferrotungsten is reduction of ferberite or some other concentrate in the electric furnace, with carbon as a reducing agent, followed by a subsequent refining and decarburization. Reduction with carbon takes place according to the following reaction:

$$FeWO_4+4C=FeW+4CO.$$

Theoretically the product would contain 76.6 per cent tungsten. The slag may contain as high as 8 per cent FeO, but less than 1 per cent WO_3. Theoretically the reduction of 100 parts of alloy from 126 parts of ferberite requires 20 parts of carbon, but practically about 25 per cent excess of carbon is charged. Small amounts of lime and fluorspar are used to flux the silica. Operating in this manner with an excess of carbon a product is made containing 3 per cent carbon, 70 per cent tungsten, 0.05 per cent phosphorus, and 0.01 per cent sulphur; the slag contains below 1 per cent WO_3. If desired, this product can be tapped from the furnace, but as its behavior is erratic, it is preferable to allow the metal to collect in the bottom of the furnace, forming, in a 150-kw. furnace, a button about 3 feet in diameter, 6 inches thick, and weighing 1,000 to 1,200 pounds; the slag is poured off by tilting as it fills the furnace.

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[19] Keeney, R. M., The manufacture of ferroalloys in Colorado: Eng. and Min. Jour., vol. 106, No. 9, Aug. 31, 1918, p. 405.

Smelting of ferberite concentrate.—A typical operation is conducted as follows: The initial charge is 65 pounds of a mixture composed of 200 pounds concentrate, 42 pounds coke, 56 pounds lime, 6 pounds fluorspar. Three more 65-pound charges are added at intervals of half an hour, and at 2½ hours from the start the furnace is tilted and the slag poured. This cycle is repeated until a 1,200-pound button has been formed, requiring 24 to 36 hours. The furnace is allowed to cool, is torn down, and the button of metal removed. This is then cleaned and broken up, the breaking process being somewhat difficult with a 3 per cent carbon alloy.

The slags produced average 0.72 per cent tungsten, and though there is considerable mechanical loss, due principally to dusting, the total loss in practice does not exceed 5 per cent.

Refining ferrotungsten.—The crude metal, broken to about 6-inch size, is refined as follows: A charge of 150-pound metal and 75-pound ferberite concentrate is smelted for half an hour, when 12 pounds of fluorspar is added. After another 3 hours the slag is poured, and a fresh charge is started. The process is continued for from 36 to 48 hours, until a button weighing 1,500 pounds has formed in the furnace, a larger button being permissible because the low-carbon alloy is easier to break. The furnace is allowed to cool, is torn down, and the button of metal removed, cleaned, and broken. The refined button forms compactly and is free from slag. The refining reduces the carbon from 3 to 0.8 per cent, the phosphorous from 0.05 to 0.01 per cent, leaves the sulphur the same, and increases the tungsten from 70 to 75 per cent. The refining slag contains 5 to 20 per cent tungsten, and is resmelted in a special run.

Ferrotungsten is one of the easiest ferroalloys to manufacture, except for the fact that it must be made in a knockdown furnace, due to the high melting point of the alloy, about 2,500° C. Metallurgically it is simple, as tungsten has not so great a tendency to form carbides or to oxidize as have chromium and uranium. Ferrotungsten containing less than 1 per cent carbon can be made in a single smelting operation by careful regulation of the carbon in the charge and the use of an acid slag. The disadvantage of this method is that the product is less pure, and the slag loss is considerably higher. The metal will contain much higher phosphorus and sulphur, and the total loss of tungsten may be 25 per cent, as compared with 10 to 15 per cent by the two-stage process.

Nearly every manufacturer of ferrotungsten has tried to tap it from the furnace, but none has continued the practice. If the tungsten is reduced to 55 to 60 per cent, the alloy can be tapped, even with carbon as low as 0.2 per cent. With tungsten at 70 per cent, metal containing 0.9 per cent carbon has run out while pouring slag, but generally any metal which comes out with the slag will contain 1.25 per cent carbon. The irregularity of tapping operations with ferrotungsten has prevented the adoption of that method, because if part of the metal must be retained in the furnace, it is cheaper to keep it all there. Furthermore, the poured metal is very hard to break.

In ferrotungsten production it is essential to maintain regularity of charging, as this is the only way to insure homogeneous composition of the metal, as the button is not melted throughout. Even during refining, the metal does not melt to any extent, the action taking place while the metal is in a pasty condition. To insure a more homogeneous product, consumers require the alloy to be crushed to one-fourth inch, this crushing forming one of the main trials of the manufacturer.

Ferrotungsten analyses.

	Per cent.
Tungsten	69.7 –74.19
Carbon	.88 – 1.00
Silicon	.39 – .76
Manganese	.15 – .53
Sulphur	.002– .055
Phosphorus	.013– .037

Analyses of ferrotungsten made from ferberite by smelting and refining indicate the limits in composition shown in the table.

IMPURITIES IN FERROTUNGSTEN MANUFACTURE.

The most objectionable impurities in concentrates used for the manufacture of ferrotungsten are phosphorus, copper, tin, and sulphur. Of these impurities the only one that ferberite concentrate is likely to contain to an injurious degree is phosphorus. Some phosphorus may be removed by magnetic concentration, or from crude ferrotungsten, by basic refining. The general guarantee made by ferrotungsten manufacturers is that the phosphorus in the ferrotungsten will not exceed the percentage of phosphorus in the concentrate smelted. Guarantees made by the two companies named below were as follows:

Quality of ferrotungsten as guaranteed.

	Vanadium Alloy Steel Co.	Ferro-Alloy Co.
	Per cent.	*Per cent.*
Tungsten	70–80	70–80
Carbon, not over	.5	1
Manganese, not over		.5
Silicon, not over		1
Sulphur, not over	.05	.05
Phosphorus, not over	.08	.05
Copper		Trace.

The usual guarantee for carbon is now that the metal will contain less than 0.5 per cent. With an acid slag the percentage of carbon can be kept this low by a single operation, but with a basic slag the metal first produced must be refined. In smelting ferberite concentrate no difficulty is experienced in keeping the manganese in the ferrotungsten less than 0.5 per cent, but in smelting wolframite an acid slag must be used to cause the manganese to pass into the slag. Silicon seldom exceeds 1 per cent unless there is an excess of reducing agent in the charge. With a basic slag the sulphur usually runs less than 0.01 per cent, but with an acid slag it is easily kept below 0.05 per cent in the ferrotungsten. Phosphorus tends to be higher in the ferrotungsten produced by the single-stage acid process than in that made by the double-stage basic process, in which the phosphorus is removed as in the basic refining of steel. The percentage of copper in ferrotungsten is generally equal to that in the concentrate.

METHOD OF SAMPLING AND ANALYSIS OF FERROTUNGSTEN AND TUNGSTEN METAL POWDER.

The method of sampling and analyzing ferrotungsten and tungsten metal powder at the laboratories of the Tungsten Products Co., at Boulder, Colo., have been described especially for this bulletin by A. J. Guerber and F. J. Gibson, as follows:

PREPARATION OF SAMPLE.

The refined ferrotungsten, crushed and rolled to pass a ½-inch mesh screen, goes to a machine which automatically mixes, samples, and sacks the product for shipment. The sampler, which is of a special type developed by this company, cuts one-fifth of the total mix. This cut passes through the machine again after being rolled to approximately ¼-inch mesh. If the original mix is very large, the operations is repeated to bring the weight of the sample down to less than 500 pounds. This final sample from the automatic is passed repeatedly through a riffle sampler until the last cut weighs 15 to 50 pounds. This goes to the laboratory sampling department, where it is reduced in a steel mortar to pass a 1/16-inch screen. The pounding necessary for this process is done by a 2-inch Ingersoll-Rand piston drill set up vertically in a frame above the mortar and pestle in such a manner that it is used as an air hammer, as shown by Plate XV, *A* (p. 43). The sample is thoroughly mixed and split twice, and one-quarter is then further reduced to pass a 10-mesh screen. This procedure of screening, mixing, and splitting is continued through a series of screens until the last portion passes the 200-mesh screen, giving the analyst about 100 grams of finely powdered metal. By this system the reduction of a 20-pound sample received from the smelter to a 100-gram laboratory pulp of 200-mesh powder requires about one hour.

DETERMINATION OF TUNGSTEN.

Mix two 1-gram samples separately in nickel crucibles with about 5 grams of sodium peroxide and a cover of sodium carbonate and fuse in an electric muffle at about 800° C.; it requires six to seven minutes to make a perfect decomposition. Cool, leach with water in covered 400-c. c. beakers. Remove crucibles and acidify carefully with hydrochloric acid, using moderate excess. Add about 2 c. c. nitric acid or H_2O_2, to assist in dissolving any small fragments of nickel oxide scale, and bring to a boil. Most of the tungsten separates as H_2WO_4 and all iron, nickel, and salts are in solution. Add about 40 c. c. cinchonine solution, stir and let stand in a warm place for about four hours, or preferably over night. This precipitates all the remaining tungsten as cinchonine tungstate, and has been proved as effective a reagent for this purpose as quinine hydrochloride.

Filter off the settled precipitate through a 15-cm. close-weave paper over a platinum cone and suction flask. Wash thoroughly with warm dilute cinchonine wash water, using gentle suction. The precipitate will not stick to the beaker if a few drops of hydrofluoric acid are used when copping out. The washed precipitate is transferred to a shallow gold dish of known weight and ignited to WO_3 in the muffle. Cool, moisten with 10 or 12 drops of hydrofluoric acid to remove any traces of SiO_2 which might be carried down, dry and ignite again. Weigh and calculate to W. The factor is 0.793. Duplicate determinations carried out in this way should check to within 0.10 per cent.

Cinchonine solution is prepared by dissolving 50 grams of cinchonine alkaloid in 2 liters of cold water containing 150 c. c. of hydrochloric acid.

Cinchonine wash water, made from 100 c. c. of the above cinchonine solution and 50 c. c. hydrochloric acid diluted to 1 liter with hot water and used from a bulb wash-bottle, is convenient.

DETERMINATION OF CARBON.

As ferrotungsten and tungsten metal powder burn completely and readily in oxygen, carbon is easily determined in any type of combustion train. In this laboratory the preferred method is to catch the CO_2 in a Meyer bulb in 2 per cent barium hydrate solution, filter, wash, and weight the $BaCO_3$.

DETERMINATION OF SULPHUR.

The usual fusion-oxidation methods of determining sulphur are open to objections when applied to a ferroalloy or other material whose sulphur content is below 0.05 per cent. The blank is always high, due to traces of sulphates in fluxes and reagents used and the danger of picking up fumes or traces of sulphuric acid from the apparatus of a general laboratory.

We have found that the sulphur content of ferrotungsten occurs as sulphides which, in a finely ground sample, are completely decomposed, yielding H_2S. This suggests at once the application of the evolution method, which has been used successfully in this laboratory for this purpose for two years.

Two to five grams of 200-mesh ferrotungsten or metal powder are weighted into an evolution flask. Cover with 50 c. c. water and add 25 c. c. strong hydrochloric acid by way of the separatory funnel. Heat to boiling and boil for about 10 minutes, catching the evolved gas, air, and steam in ammoniacal cadium chloride solution. The usual yellow precipitate of cadmium sulphide shows the presence of sulphides in the alloy. Wash the cadmium chloride solution from the bulbs, then cool, acidify with hydrochloric acid, and titrate at once against a weak iodine solution (1 c. c.=0.0005 g. sulphur), using starch as an indicator. The iodine solution is standardized by using a Government standard steel of known sulphur content, or by using a ferrotungsten of known sulphur content.

Boiling with dilute hydrochloric acid does not dissolve the alloy, but does decompose the sulphides present, as may be shown by filtering off the contents of the evolution flask after the operation and examining for sulphur by a fusion method or by a combustion method.

DETERMINATION OF PHOSPHORUS.

The removal of phosphorus from the alkaline liquor after a fusion decomposition of ferrotungsten presents certain difficulties. Precipitation with magnesia mixture from dilute solutions is incomplete unless much time is allowed for this step, also precipitation from concentrated solutions either as magnesium ammonium phosphate or as aluminium phosphate tends to drag down some tungsten, involving another step for its removal. To avoid these difficulties we have introduced the use of uranium acetate at this stage. The precipitation is quantitative from moderately dilute solutions and no dragging down of tungsten occurs. The details of procedure described in the following paragraphs are the result of considerable research and embody the experience of many hundreds of determinations made in this laboratory.

Weigh 1 gram sample into a nickel crucible containing 6 to 8 grams c. p. Na_2O_2. Mix thoroughly and cover with about 1 gram of c, p. Na_2CO_3. Ignite in a muffle to complete fusion of the fluxes. Cool and leach out carefully with warm water in a 250-c. c. covered beaker. Remove crucible and partly neutralize the NaOH with 25 c. c. HCl (1–3) to permit filtering without dilution. Stir and then allow the dense $Fe(OH)_3$ to settle out while standing on a warm plate.

Decant through a 12½-cm. qualitative paper into a 600-c. c. beaker. Wash precipitate into the filter, allow it to drain thoroughly, then give it two good washes with hot water, stirring up the precipitate well with a fine jet. Reserve this precipitate of $FE(OH)_3$, which contains a small amount of phosphorus, to add to the uranium precipitate obtained from the tungsten solution.

Acidify the filtrate of sodium tungstate and sodium phosphate with acetic acid, using 30 c. c. of 40 per cent acetic. Add 2 c. c. of 5 per cent uranium acetate solution. Boil off all CO_2. Make just alkaline with NH_4OH (1–1). This will require about 20 c. c. The precipitate is ammonium uranyl phosphate. Add 2 c. c. more uranium acetate, which precipitates at once as uranium hydroxide and serves to drag down the last traces of phosphate as well as to assist in filtering the ammonium uranyl phosphate. Boil off any large excess of NH_4OH. Filter through a qualitative paper and wash twice with hot water to remove tungsten. Place the precipitate in the same beaker with the iron precipitate obtained above, add 30 c. c. water and 25 c. c. strong nitric acid. Boil until both precipitates are in solution and the filters reduced to pulp. Filter into a 500-c. c. Erlenmeyer flask, washing the pulp once with hot water. Add 5 c. c. of 5 per cent $KMnO_4$ solution to acid filtrate. Boil to oxidize all phosphorus to H_3PO_4. Clear of MnO_2 by adding 2 c. c. H_2O_2 (1–1). Boil to remove excess H_2O_2 and cool under the tap. Add 45 c. c. NH_4OH (1–1), which will nearly neutralize the nitric acid, then 50 c. c. of molybdate solution. Stopper the flask and shake for five minutes, and let stand at least 15 minutes before filtering. Filter through a 9-cm. filter. Wash with acid ammonium sulphate, wash solution to remove all nitromolybdate.

Dissolve the yellow phosphomolybdate in hot dilute ammonia water, which must be free from chlorides, into the same Erlenmeyer flask. Wash the paper well with alternate washes of ammonia water and hot water. When cool, acidify with 5 c. c. of H_2SO_4 (1–1) and pass through the reductor, following with a thorough wash of cold water. Titrate against standard $KMnO_4$ solution to the same end point as used in determining the reductor blank.

The iron value of the standard permanganate solution times 0.0163 equals the P. value * * *. The ammonium sulphate wash solution must be distinctly acid. A good formula is:

H_2SO_4 (1–1), 100 c. c.; NH_4OH (1–1), 60 c. c.; water, 2 liters.

If convenient, it is well to allow the precipitate of yellow phosphomolybdate to stand about 1 hour after shaking for 2 minutes.

A blank on all reagents, including a fusion of Na_2O_2 and Na_2CO_3, should be carried through with the determinations. It is sometimes desirable to add a measured quantity of dilute phosphate solution of known P. content to the blank for close work.

Checks should agree within 0.015 per cent P.

The calculation of the P. factor is described by Lord and Demorest.[20] The composition of ammonium phosphomolybdate is expressed by the formula $(NH_4)_3PO_4.12MoO_3$, which shows that 1 P. is equivalent to 12 Mo, or 36 Fe.

Much has been written and many discussions have taken place in the past concerning the state of reduction of a molybdenum solution when passed through a Jones reductor. This controversy has been considered in detail in Randall's [21] article, and it has been proved conclusively that on passing through the reductor the molybdenum is reduced to Mo_2O_3 or $Mo_{12}O_{18}$.

The factor of 0.0163, described by Lord and Demorest, assumes that the MoO_3 solution is reduced to $Mo_{12}O_{19}$ rather than $Mo_{12}O_{18}$, as described by Randall. Therefore, in assuming the reduction as $Mo_{12}O_{19}$, one atom of P equals 34 atoms of Fe, whereas in using $Mo_{12}O_{18}$, one atom of P equals 36 atoms of Fe or 12 atoms of Mo, as previously described, thus making the factor 0.0154 in place of 0.0163. The difference, however, that would result from using the factor 0.0154 would be so small that it could, for ordinary analysis, be neglected. Assuming a normal permanganate solution, this difference would for 1 c. c. amount to 0.000058 gram P; for tenth or twentieth normal, which would be the strength most likely used, this figure would be correspondingly diminished.

This uncertainty as to the reduction factor, etc., can be overcome if the permanganate solution is standardized against a standard steel ore or pig iron, of known phosphorus content, which should be treated by the method used in the regular analysis.

BUYING OF TUNGSTEN ORES.

The buying specifications of tungsten ores have been summarized by Young,[22] as follows:

It is unfortunate that the buying of tungsten ores in accordance with definite grades and specifications has not been standardized. Changing specifications and uncertainty respecting the marketability of tungsten ores and concentrates have made it difficult for some producers to satisfy themselves that they have received a fair price for their product. Prices are made on the basis of per unit of tungstic acid on the short ton. Scheelite and wolframite are recognized as two important types, although there is little distinction in price of either type where the accompanying impurities do not exceed the following amounts: Sulphur, not over 0.1 per cent; copper, not over 0.02 per cent; tin, not over 0.02 per cent; phosphorus, not over 0.02 per cent. Ores of this type are termed pure or high-grade ores, and where they exceed 60 to 65 per cent WO_3 or over they command the highest prices. The scheelite concentrate and ore of the Atolia mine are in a separate class and also command a high price. All other ores, whether scheelite, wolframite, or otherwise, are classed as impure ores, and are sold at various prices, depending on the amount and kind of impurity as well as upon the facilities of the buyer for the treatment of the different kinds of ore.

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[20] Lord, N. W., and Demorest, D. J., Metallurgical analysis, 4th ed., 1916, p. 50.

[21] Randall, D. L., The behavior of molybdic acid in the zinc reductor: Am. Jour. Sci., ser. 4, vol. 24, 1907, p. 313.

[22] Young, G. J., The tungsten industry in 1918: Eng. and Min. Jour., vol. 107, Jan. 11, 1919, p. 78.

That the limits given are not universally recognized is shown by Charles Hardy's classification, which places the maximum percentage of impurity for high-grade ore and concentrate at: Tin, 0.05 per cent; copper, 0.05 per cent; and phosphorus, 0.04 per cent. All ores and concentrates exceeding these amounts of impurity are called off-grade ores. Other firms have specifications which make no use of the terms high-grade, off-grade, or impure, but merely specify the lower limit of tungstic acid and the upper limit of certain impurities. The highest-grade wolframite marketed contains 70 per cent tungstic acid, traces of tin and copper, and manganese under 1 per cent. Tungstic acid content ranges from 10 to 72 per cent, although there is recognized, to some extent at least, a base grade containing 65 per cent tungstic acid. In some cases a 60 per cent grade is named. There is no recognized deduction or addition in price for a variation in tungstic acid content in either direction from the base grade. Ores with impurities may contain the followng range: Sulphur, up to 3 or 4 per cent; tin, up to 10 per cent; copper, up to 3 per cent; phosphorus, up to 0.3 per cent. Certain foreign ores even exceed these percentages, as the tungstic acid content may reach 45 per cent; copper, 6 per cent; and tin, 19 per cent. Some buyers handle only scheelite, others wolframite, and still others both types. There is need for a clean-cut classification.

Hess,[28] in summing up the quality of ore demanded by users, says:

> Most of the processes used for reducing tungsten from its ores also partly or wholly reduce nearly all the metallic and some other impurities in the ores, and these impurities are carried with the tungsten into the steel to which it is added. For such use iron makes no difference, but a number of other elements are not wanted, either because, like copper and phosphorus, they are detrimental to the steel or because, like manganese, if they are wished in the steel they can be added more advantageously in some other way. Objectionable impurities found in tungsten ores are antimony, arsenic, bismuth, copper, lead, manganese, nickel, tin, zinc, phosphorus, and sulphur. Few of these occur in large quantity in ores found in this country. Copper is perhaps the commonest hurtful impurity, and therefore most is said about it, but ores from some foreign countries contain nearly all the impurities mentioned. During the early part of 1916 tungsten ores were so eagerly sought that nearly all offered were bought with little objection to impurities, but under more normal conditions consumers are much more particular.
>
> The wet chemical processes give more opportunity to get rid of most impurities than the electrolytic process, so that companies using wet chemical processes are, as a rule, though not uniformly, least particular about the ores they buy. Two of the firms that use wet chemical processes buy tungsten ores

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[28] Hess, F. L., Tungsten minerals and deposits: U. S. Geol. Survey Bull. 652, 1917, p. 74.

almost without regard to the impurities present, but one objects to more than 2 per cent copper, and both buy ores containing as little as 20 per cent WO_3.

Only the one firm mentioned is known that does not object to copper in any grade of ore. Another will take cupriferous ore if the content of WO_3 is sufficiently high. The others either will not take copper-bearing ore when other ores are to be obtained or set limits of 0.2 to 2 per cent copper and not less than 50 per cent WO_3, except that one firm will take ores that carry 5 per cent or more copper, for such a percentage will pay for separation.

Two companies take ores without regard to impurities other than copper, provided the content of WO_3 is sufficiently high. Most of the companies object to tin, sulphur, phosphorus, antimony, arsenic, bismuth, lead, and zinc; two of them object to manganese; and one to nickel. The last company referred to set extreme limits of 0.25 per cent for phosphorus, 0.25 per cent for nickel, 6 per cent for manganese, and a trace of arsenic.

ORE SCHEDULE.

A typical ore-schedule sheet follows:

TABLE 8.—*A typical ore-schedule sheet.*

[Schedule of prices for tungsten ore delivered at mill. Subject to change without notice. No sampling and assaying on lots over $50 in value. In effect with tungsten at about $18 per unit, f. o. b. New York.]

	Per pound.	Per ton.		Per pound.	Per ton.
Tungstic acid:	*Cents.*		Tungstic acid—Continued.	*Cents.*	
2 per cent	12	$4.80	32 per cent	47.4	$303.60
3 per cent	15	9.00	33 per cent	47.4	313.40
4 per cent	22	17.60	34 per cent	47.5	323.20
5 per cent	31	31.00	35 per cent	47.6	333.00
6 per cent	36	43.20	36 per cent	47.6	342.80
7 per cent	40	56.00	37 per cent	47.6	352.60
8 per cent	42	67.20	38 per cent	47.6	362.40
9 per cent	43	77.40	39 per cent	47.7	372.20
10 per cent	43.5	87.00	40 per cent	47.7	382.00
11 per cent	44	96.80	41 per cent	51.2	420.25
12 per cent	44.7	107.40	42 per cent	52.5	441.00
13 per cent	45.2	117.60	43 per cent	53.7	462.25
14 per cent	45.4	127.20	44 per cent	55	484.00
15 per cent	45.6	137.00	45 per cent	56.1	506.25
16 per cent	45.6	146.00	46 per cent	57.5	529.00
17 per cent	46	156.60	47 per cent	58.7	552.25
18 per cent	46.2	166.40	48 per cent	60	576.00
19 per cent	46.3	176.20	49 per cent	61.2	600.25
20 per cent	46.5	186.00	50 per cent	62.5	625.00
21 per cent	46.6	195.80	51 per cent	63.7	650.00
22 per cent	46.7	205.60	52 per cent	65	676.00
23 per cent	46.8	215.40	53 per cent	66.5	702.25
24 per cent	46.9	225.20	54 per cent	67.5	729.00
25 per cent	47	235.00	55 per cent	67.5	756.00
26 per cent	47	244.80	56 per cent	70	784.00
27 per cent	47.1	254.60	57 per cent	71.6	812.25
28 per cent	47.2	264.40	58 per cent	72.4	841.00
29 per cent	47.2	274.20	59 per cent	73.7	870.00
30 per cent	47.3	284.00	60 per cent	75	900.00
31 per cent	47.4	293.80			

PRODUCTION OF TUNGSTEN CONCENTRATES IN THE BOULDER DISTRICT, IN THE UNITED STATES, AND IN THE WORLD.

The production of tungsten concentrates in Colorado has been, as shown in Table 9 below, taken from the Fifteenth Biennial Report of the Colorado Bureau of Mines, 1917–18, page 121:

TABLE 9.—*Tungsten production of Colorado.*

	Tons.	Price per unit.	Total value.
1900. High-grade ore, 63 per cent WO_3	40	$1.30	$3,216
1901. High-grade ore and concentrates averaging 65 per cent WO_3	65	2.25	8,875
1902. Ores and concentrates	166	2.50	24,900
1903. Mainly concentrates	243	2.50	36,317
1904. Ores and concentrates averaging 55 per cent WO_3	375	5.50	125,000
1905. Mainly concentrates	642	6.00	231,120
1906. Mainly concentrates	789	6.54	309,608
1907. Mainly concentrates	1,146	8.83	573,643
1908. High-grade ore 180 Concentrates 407	587	....	164,220
1909. Production on basis of 60 per cent WO_3	993	$5.00- 9.00	391,160
1910. Production on basis of 60 per cent WO_3	1,221	6.50- 8.50	553,100
1911. Production on basis of 60 per cent WO_3	730	4.50- 8.50	261,492
1912. Production on basis of 60 per cent WO_3	775	5.60- 7.50	293,611
1913. Production on basis of 60 per cent WO_3	953	7.50	428,726
1914. Production on basis of 60 per cent WO_3	630	5.85- 9.00	262,000
1915. Production on basis of 60 per cent WO_3	960	5.50-45.00	1,688,640
1916. Production on basis of 60 per cent WO_3	2,401	34.15	4,919,649
1917. Production on basis of 60 per cent WO_3	2,707	20.00	3,248,400
1918. Production on basis of 60 per cent WO_3	1,910	16.00	1,833,600

Nearly all of Colorado's production was from the Boulder district. It was in 1898 that wolfram ores were first reported found at various places in Colorado, but no shipments were made.[24] The first shipment reported from Colorado was in 1899, when Poulot and Voilleque mined 3 tons of ore; the location, however, was not given.[25]

In the following discussion concerning production, all calculations are based on short tons containing 60 per cent WO_3 unless otherwise stated. Statistics and questions concerning the quantities produced in the United States by districts and States, world production by countries, imports and exports, uses, prices, mining conditions, etc., of tungsten are found in "Mineral Resources of the United States," published annually by the United States Geological Survey; also in "The Mineral Industry," published annually by the McGraw-Hill Co., of New York. Besides these two principal sources of information, reports from the State bureaus of mines or mining bureaus and the current technical journals furnish data.

In 1900 the first important shipments of tungsten concentrates were made, and since then Colorado has ranked as the chief producing State.

BOULDER COUNTY PRODUCTION FOR PERIOD 1914-1919, AS COMPARED TO THAT OF CALIFORNIA.

According to Mineral Resources for 1914, the production of the Atolia field in the Mojave Desert, Calif., for that year exceeded that of Boulder County, Colo., for the first time. It was claimed that Boulder County produced 467 tons out of the country's total production of 990 tons. The table as given above for Colorado's production represents 630 tons produced in Colorado for 1914, which practically all came from the Boulder field.

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[24] The Mineral Industry, 1898, vol. 7, p. 719.

[25] The Mineral Industry, 1899, vol. 8, p. 632.

In 1915 Colorado's production, 963 tons, was practically equal to the production of California, 962 tons; also, according to Hess, the total production of the United States for 1915 was 2,332 tons.[26]

The production of tungsten ores in the United States for 1916 was the largest ever made by any country in the world, the amount being 5,923 tons, valued at $12,074,000; Colorado again exceeded California, producing 2,401 tons against California's 2,171 tons.[27]

As Mineral Resources for 1917 was not available when this report was written, the data for 1917 has been taken from the Mineral Industry, 1917, vol. 26, p. 699. California is given as the largest tungsten-producing State; in fact, as the largest scheelite producer of the world. Production for 1917 is given as 2,250 tons against 2,200 tons for Colorado. The United States produced 5,313 short tons. According to the tungsten production table of Colorado, as given above, Colorado produced in 1917, 2,707 tons; and, according to the report of the California State Mining Bureau, 2,458 tons were produced in California.[28]

The production of tungsten ores in the United States for 1918, as given by Hess,[29] was less than that of the preceding two years, and was 5,065 short tons of concentrates, carrying 60 per cent tungsten trioxide. The largest producing State was Colorado, with 1,910 tons of ferberite. California produced 1,781 tons of scheelite, and Nevada 885 tons of scheelite. According to the California State Mining Bureau, under date of August 13, 1919, the production of concentrates in 1918 was 1,982 tons.

The production for 1919 for the Boulder district can be but roughly estimated, for during the early part of the year, after the price of tungsten dropped to around $13 a unit, practically all the mills closed, and some went completely out of business. The two largest producers, the Wolf Tongue and the Primos, closed their mills completely about March. The probable production for 1919 has been placed at 500 tons of concentrates. Production in August, 1919, was practically at a complete standstill, as tungsten concentrates (60 per cent WO_3) were selling at $7 to $10 a unit. The only mill found in operation during a brief visit by the writers in August, 1919, was the Vasco at Tungsten, Colo., which was producing about 20 tons of tungsten concentrates per month. The Red Sign mill at Ferberite had been completely dismantled and the equipment moved out of the district.

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[26] Hess, F. L., Tungsten: Mineral Resources of the United States, 1915, p. 823.

[27] Hess, F. L., Tungsten: Mineral Resources of the United States, 1916, p. 789.

[28] Tungsten, molybdenum, and vanadium, Report No. 4, March, 1918, p. 9.

[29] Hess, F. L., Tungsten ore production in 1918: Eng. and Min. Jour., vol. 107, Feb. 8, 1919, p. 285.

The two largest producing mines of California—the Atolia and the Tungsten Mines Co.—closed down in March, 1919, and September, 1919, respectively. The mines in the Nevada field closed down after operating only a few months.

The summary of the tungsten market situation in 1919[30] and the tungsten mining industry in 1919[31] has been given in the annual review number of the Engineering and Mining Journal.

Little tungsten ore was produced in the United States in 1919, and a vast stock accumulated principally from the Far East that sold for around $7 a unit, which caused the American mines to close.

Figures, as given in the Mineral Resources, give production for California and Colorado in 1919 at 277 short tons of 60 per cent WO_3 concentrates valued at $323,799.

PRODUCTION OF TUNGSTEN ORES IN THE UNITED STATES.

The production of tungsten ores in the United States has been, as shown in Table 10, taken from Mineral Resources.[32]

Production has been recorded by Mineral Resources, United States Geological Survey, only since 1900. Before that time it was insignificant. In 1897 there was no production of tungsten ores in the United States; the demands did not exceed 70 to 80 tons per annum.[33] The amounts produced in 1898 and 1899 were negligible.

TABLE 10.—*Concentrated tungsten ores produced in the United States, 1900–1919.*

Year.	Quantity (short tons).	Value.	Year.	Quantity (short tons).	Value.
1900	46	$11,040	1910	1,821	$832,992
1901	179	27,720	1911	1,139	407,985
1902	184	34,040	1912	1,330	502,158
1903	292	43,639	1913	1,537	672,118
1904	740	184,000	1914	990	435,000
1905	803	268,676	1915	2,332	4,100,000
1906	928	348,867	1916	5,923	12,074,000
1907	1,640	890,048	1917	6,144	6,783,000
1908	671	229,955	1918	5,061	7,049,300
1909	1,619	614,370	1919	327	353,900

The production of tungsten ores in this country from year to year can be fairly compared in the table only since and beginning with

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[30] Hardy, Charles, The tungsten market situation in 1919: Eng. and Min. Jour., vol. 109, No. 3, Jan. 17, 1920, p. 210.

[31] Young, George J., The tungsten mining industry in 1919: Eng. and Min. Jour., vol. 109, No. 3, Jan. 17, 1920, p. 211.

[32] Hess, F. L., Tungsten: Mineral Resources of the United States, 1916, p. 789.

[33] Mineral Industry, 1897, vol. 6, p. 651.

1906, as before that date no effort was made to reduce the ores to a common basis of tungsten content. Since 1906 the ores have been, as far as possible, reduced to an equivalent of ore carrying 60 per cent of tungsten trioxide (WO_3), which is the ordinary commercial basis in the United States.

According to Frank L. Hess,[34] not since 1902 has the United States produced so small a quantity of tungsten concentrates as in 1920. The Wolf Tongue Mining Co. and the Vasco Mining Co., of Boulder, Colo., were the only American tungsten mines. They produced an equivalent of 216 short tons of ferberite ore carrying 60 per cent tungsten trioxide (WO_3). The large stock of tungsten ores on hand carried over from 1919, and the continued importation of tungsten ores from China and South America, coupled with the low price a unit, between $7 and $3 for 60 per cent grade, kept the American producing mines closed. The prices for 1920 and continuing in 1921, at $3 a unit, have never been so low in the entire history of the tungsten industry, except previous to 1904, as shown in Table 10.

COST OF PRODUCTION IN BOULDER COUNTY DISTRICT.

Before the war the average cost of production for the Boulder district had been conservatively placed at $11 to $12 a unit. In some of the small mills treating low-grade ore the cost had exceeded these figures. In April, 1916, tungsten ore sold at $75 to $85 a unit in New York. In May, 1916, the price dropped to a nominal quotation of $40 a unit. In September, 1916, tungsten ore could be freely purchased for about $17 a unit. During the fall of 1916—when tungsten was selling for that price—several of the smaller tungsten mills in the district closed down, and it was stated then that the marginal profits were too small and would not warrant continuing operations. The largest producers, however, continued work, for, beginning with the summer of 1917 and to the signing of the armistice in November, 1918, the market for tungsten ore remained practically steady, around $20 a unit, enough to warrant a fair margin of profit.

Mining conditions have considerably changed from what they were in 1916 and 1917. Then the ore as soon as it was discovered, was gutted out; practically all the easily accessible ore in sight was quickly mined and very little systematic development was done. Now, the situation at the mines is different; with no easily accessible ore in sight, considerable new development and opening of new ore bodies must be done, and consequently this, coupled with the increased salaries demanded by the miners, has greatly raised the cost of production.

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Table 11 summarizes the total production of tungsten ore, 1905 1916, according to Hess.[35] For the detailed productions in the fferent countries the reader is referred to Mineral Resources or ineral Industry.

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stimated as far as possible in short tons of concentrate containing 60 per cent tungsten trioxide.]

ite.	Tons.	Date.	Tons.
05[a]	4,026	1913	8,864
06[a]	4,367	1914	7,813
07	6,137	1915	11,840
08	4,118	1916	23,360
09	5,774	1917	25,523
10	7,570	1918	31,865
11	7,517	1919	12,000
12	9,710		

[a] Mineral Resources for 1914, p. 938.

During the period shown the United States was the chief producer, xcepting in 1912, 1913, 1914, and 1915, when Burma and the Shan tates were the chief producers.

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> The present largest producer outside of the United States is China. Tungsten res were found in China in 1917—that is, the first production that amounted

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[36] Kappel, H., Chinese tungsten and the American market: Eng. and Min. Jour., vol. 11, No. 7, 1921, p. 308.

[37] Hess, F. L., Tungsten production analyzed: Jour. Am. Min. Congress, July, 1919, . 231.

1906, as before that date no effort was made to reduce the ores to a common basis of tungsten content. Since 1906 the ores have been, as far as possible, reduced to an equivalent of ore carrying 60 per cent of tungsten trioxide (WO_3), which is the ordinary commercial basis in the United States.

According to Frank L. Hess,[34] not since 1902 has the United States produced so small a quantity of tungsten concentrates as in 1920. The Wolf Tongue Mining Co. and the Vasco Mining Co., of Boulder, Colo., were the only American tungsten mines. They produced an equivalent of 216 short tons of ferberite ore carrying 60 per cent tungsten trioxide (WO_3). The large stock of tungsten ores on hand carried over from 1919, and the continued importation of tungsten ores from China and South America, coupled with the low price a unit, between $7 and $3 for 60 per cent grade, kept the American producing mines closed. The prices for 1920 and continuing in 1921, at $3 a unit, have never been so low in the entire history of the tungsten industry, except previous to 1904, as shown in Table 10.

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to anything. They produced about 1,500 tons of ore. In 1918 they shipped something like 10,500 tons. The ore is found as placer stuff, where the veins have worn down so the material in the veins has been washed away, leaving this wolframite in the dirt and stream beds. There is going to be a lot of ore coming from China because they have gotten down to mining the veins.

BOLIVIA AND PORTUGAL AS PRODUCERS.

Bolivia produced in 1916, 3,624 tons, and, according to Hess,[38] is perhaps the third largest producer at the present time. Portugal is the largest producer in Europe.

UNITED STATES IMPORTS, EXPORTS, AND CONSUMPTION.

IMPORTS OF TUNGSTEN ORE, TUNGSTEN METAL, AND FERROTUNGSTEN.

Table 12 represents only in a brief form the value of the imports of tungsten ores, tungsten, and ferrotungsten in this country from 1902 to 1916; for detailed figures assigned to countries, etc., the reader is referred to the original tables given under the section "Tungsten" in Mineral Resources, or to figures compiled by the Division of Statistics of the Bureau of Foreign and Domestic Commerce.

Prior to 1908 the tungsten imports of ore into this country were in the form of ferrotungsten and tungsten metal or powder. No data has been given for the figures not given below.

TABLE 12.—*Imports of tungsten ore, tungsten metal, and ferrotungsten.*

Year.	Tons of ore (short tons).	Value.	Tons of tungsten and ferrotungsten (long tons).	Value.
1902	(a)	(a)	(a)	$7,046
1903	(a)	(a)	(a)	18,136
1904	(a)	(a)	(a)	29,439
1905	(a)	(a)	(a)	13,525
1906	(a)	(a)	(a)	....
1907	(a)	(a)	(a)	39,417
1908	(a)	(a)	(a)	488
1909	(a)	$105,506	(a)	98,205
1910	852	344,979	528.0	678,534
1911	294.5	368,618	115.07	143,802
1912	824	362,422	288	323,872
1913	401	213,122	661	835,212
1914	299	139,687	195	222,447
1915	1,534	1,044,986	7.0	9,588
1916	4,071	7,353,691	38.0	157,711
1917	4,880	4,467,608	1.0	3,606
1918 [d]	[b]10,362	11,409,237	(c)	....
1919 [d]	8,400	6,261,190	....	....

[a] Not given.

[b] First six months ended June 30, 1918. Total imports for 10 months ended Oct. 31, 1918, were 7,001 tons, valued at $7,536,875, as against 4,373 tons, valued at $3,594,961, for the corresponding period of 1917, according to the Department of Commerce.

[c] 396,460 pounds.

[d] From Mineral Industry, vol. 28, 1919. p. 689.

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[38] Hess, F. L., Work cited, p. 231.

The imports for 1920, according to Hess,[39] were equivalent to about 2,111 short tons of concentrates carrying 60 per cent WO_3. Besides the ore, 1,997,719 pounds of tungsten and ferrotungsten were imported, equivalent to about 2,250 short tons of 60 per cent ore. Hess estimates that probably more than a three years' supply is on hand, based on the average consumption before the World War.

EXPORTS.

Before the war very little was exported; so little, in fact, that the Government published no separate report. A few published figures concerning exports are as follows:

The exports of tungsten metal and alloy amounted to 2,323,811 pounds, valued at $4,122,653, in 1917, compared with 574,321 pounds, valued at $1,352,631, in 1916.[40] Exports for 1919 totaled 38,130 pounds, valued at $113,276, as compared with 1,232,260 pounds, valued at $2,545,087, in 1918.

The submarine *Deutschland*, on her second trip in December, 1916, carried back to Germany 55,125 pounds of tungsten and ferrotungsten, valued at $137,812.[41]

CONSUMPTION OF TUNGSTEN DURING 1918.

The largest consumption of tungsten in the United States was during the year 1918, as recorded by the Bureau of Mines. The following excerpt from Reports of Mineral Investigations of the United States Bureau of Mines, June, 1919, gives in detail the amount of tungsten consumed.[42]

Ten million pounds of tungsten used in 1918.—An interesting summary of the consumption of tungsten in 1918, based on questionnaires distributed by the Bureau of Mines, shows that the tungsten used in the making of tungsten powder, ferro, and acid, from May to December, 1918, amounted to about 7,500,000 pounds, from which it is estimated that the total tungsten used in such products for the calendar year of 1918 was nearly 10,000,000 pounds, and that the amount of tungsten used in making malleable products during the same time was approximately 210,000 pounds. The quantity of high-speed steel made from May to December, 1918, was nearly 30,000,000 pounds, from which it is estimated that the total weight of such steel in the year was somewhat in excess of 40,000,000 pounds; the tungsten content was properly nearly 7,000,000 pounds. The tungsten steel made from May to December, 1918, was in excess of 45,500,000 pounds, from which it is estimated that the total quantity produced in 1918 was approximately 62,000,000 pounds. The amount of 60 per cent concentrates required to make the tungsten powder, ferro, and acid used in 1918 was approximately 13,000 tons, assuming that 1 ton of 60 per cent concentrates is reduced to 760 pounds of contained tungsten in ferro.

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[39] Hess, F. L., Tungsten in 1920: Chem. and Met. Eng., vol. 24, No. 17, Apr. 27, 1921, p. 757.

[40] Fink, C. G., Mineral Industry for 1916, vol. 25, 1917, p. 722; for 1917, vol. 26, 1918, p. 690; for 1919 and 1918, vol. 28, 1920, p. 689.

[41] Hess, F. L., Nickel: Mineral Resources, 1916, U. S. Geol. Survey, 1919, p. 788.

[42] Mining Cong. Jour., vol. 5, August, 1919, p. 270; Chem. and Met. Eng., vol. 21, Aug. 15, 1919, p. 200.

The total consumption of tungsten concentrates in the United States in 1919 amounted to approximately 7,500 short tons. Consumption for 1920 is estimated at about 4,500 tons.[43]

The following diagram (Pl. XVIII) of tungsten products shows how tungsten is utilized:

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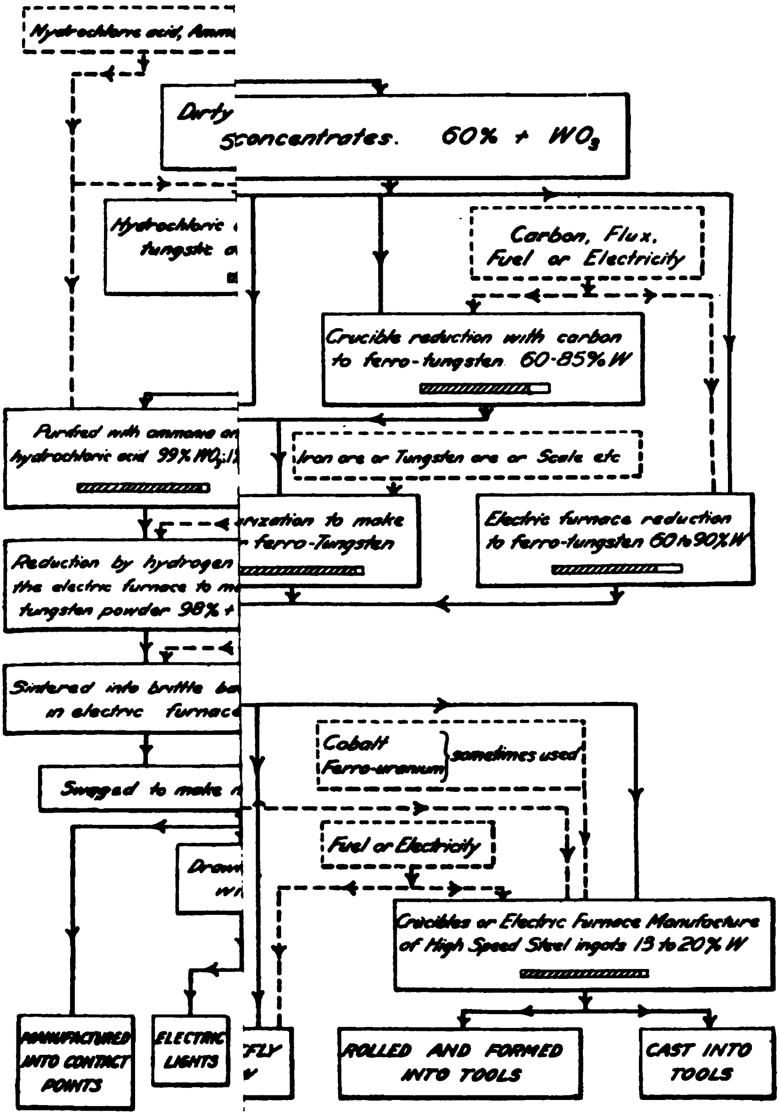
[43] Mineral Industry, vol. 28, 1919, p. 689.


Hydrochloric acid, Amm
Dirty
concentrates. 60% + WO₃
Hydrochloric
tungstic a
Carbon, Flux,
Fuel or Electricity
Crucible reduction with carbon
to ferro-tungsten 60-85% W
Purified with ammonia an
hydrochloric acid 99% WO₃:1
Iron ore or Tungsten ore or Scale etc
rization to make
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Electric furnace reduction
to ferro-tungsten 60 to 90% W
Reduction by hydrogen
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tungsten powder 98% +
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Ferro-uranium
sometimes used
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MANUFACTURED
INTO CONTACT
POINTS
ELECTRIC
LIGHTS
FLY
ROLLED AND FORMED
INTO TOOLS
CAST INTO
TOOLS

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